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UNITED STATES
DEPARTMENT OF THE INTERIOR
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BUREAU OF MINES
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INFORMATION CIRCULAR

OPERATIONS AT THE HAILE GOLD MINE, KERSHAW, S. C.

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BY

EDMUND NEWTON, D. B. GREGG, AND McHENRY MOSIER

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INFORMATION CIRCULAR

UNITED STATES DEPARTMENT OF THE INTERIOR — BUREAU OF MINES

OPERATIONS AT THE HAILE GOLD MINE, KERSHAW, S. C.[1]

By Edmund Newton[2], D. B. Gregg[3], and McHenry Mosier[4]

INTRODUCTION

The Haile mine has been the outstanding gold-mining operation and the largest single producer of gold in the Appalachian region. Gold was discovered at the Haile in 1827; previous to the present operation, production has been nearly $4,500,000 (gold at $20.67 per ounce) from somewhat over a million tons of ore, with an average extraction of about 70 percent.

The Haile property, now operated by Haile Gold Mines, Inc., is in the southern part of Lancaster County, South Carolina, 3-1/2 miles northeast of the town of Kershaw and 55 miles north of Columbia, the State capital. (See fig. 1.) The corporation has a lease and purchase contract on 1,805 acres, covering the present productive area. In addition, it holds lease contracts on 1,500 acres additional adjacent to this area.

The Haile property lies on the Piedmont Plateau near the edge of the coastal plain at an elevation of 500 to 600 feet above sea level. The hills are gently sloping and present no obstacles to economic mining.

The climate is favorable to year-round operations, as there is no prolonged excessively cold weather and seldom any snow. The summer heat is not oppressive, as the humidity is not high.

Timber can be cut on the property or can be obtained cheaply from other local sources. An ample water supply has been provided nearby through the construction of a dam on Ledbetter Creek. Plenty of both white and colored labor can be obtained from the surrounding area.

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[1] The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from Bureau of Mines Information Circular 7111."

[2] Formerly mine manager, Haile Gold Mines, Inc., and one of the Bureau of Mines consulting engineers.

[3] Superintendent, Haile Gold Mines, Inc., and one of the Bureau of Mines consulting engineers.

[4] Supervising engineer, Metal Mining Research Section, Mining Division, Bureau of Mines.

Supplies are quickly available at Kershaw and Columbia, and Charlotte. An excellent paved highway, which crosses the property near the center of present mining operations, makes it possible to reach either Columbia or Charlotte by automobile in a little more than an hour.

ACKNOWLEDGMENTS

The authors are indebted to Harlan H. Bradt, managing engineer of Haile Gold Mines, Inc., for permission to publish the details of mining and milling practice at this property. William A. Chandler, mine superintendent, M. C. Sherrill, mine accountant, and other members of the mine staff assisted in preparing data.

HISTORY

The original Haile property comprised part of the plantation of Col. Benjamin Haile, who, according to an old South Carolina district map dated 1825, had a grist mill near the site of the present mine. Published record state that gold was discovered in a creek on the property in 1827. The ear work was confined to washing the banks and bottoms of this creek or "branch but in 1829 the outcroppings of a mineralized zone were discovered, and mining of ore in place was begun.

In 1837, Cugnot, a Frenchman, built a 5-stamp mill on the property, which is reported to be the second stamp mill to be operated in the United States. Soon afterward other stamp mills were erected and, in addition, chilean mills, arrastras, and hand mortars were used.

The discovery of gold in California in 1849 attracted many miners from the Haile and other parts of the Appalachians. This situation, together with the difficulty of working at greater depths, the transition from oxidized ore to sulfides, and the effects of the Civil War, gradually led to the abandonment of operations. Little work appears to have been done until 1880, when E. Gybbon Spilsbury took charge of the property for a New York group. Shortly thereafter a 20-stamp mill was built. This operated succes fully for the first 2 years on ore from the oxidized zone above the ground water level. Thereafter increasing sulfides in the ore caused metallurgica difficulties. Many experiments were tried, but it was not until 1889 that Dr. Adolph Thies developed the Thies barrel chlorination process to recover gold from sulfides.

The Haile became the classic example of the use of chlorination for recovering gold. The property was visited by the outstanding mining and meta lurgical engineers of that time to witness operations.

From 1880 the property was operated continuously and produced an average of about 25 tons per day for the first 7 years and about 125 tons per day for the next 21 years until 1908, when a boiler explosion killed the mine manager and wrecked the 60-stamp mill.

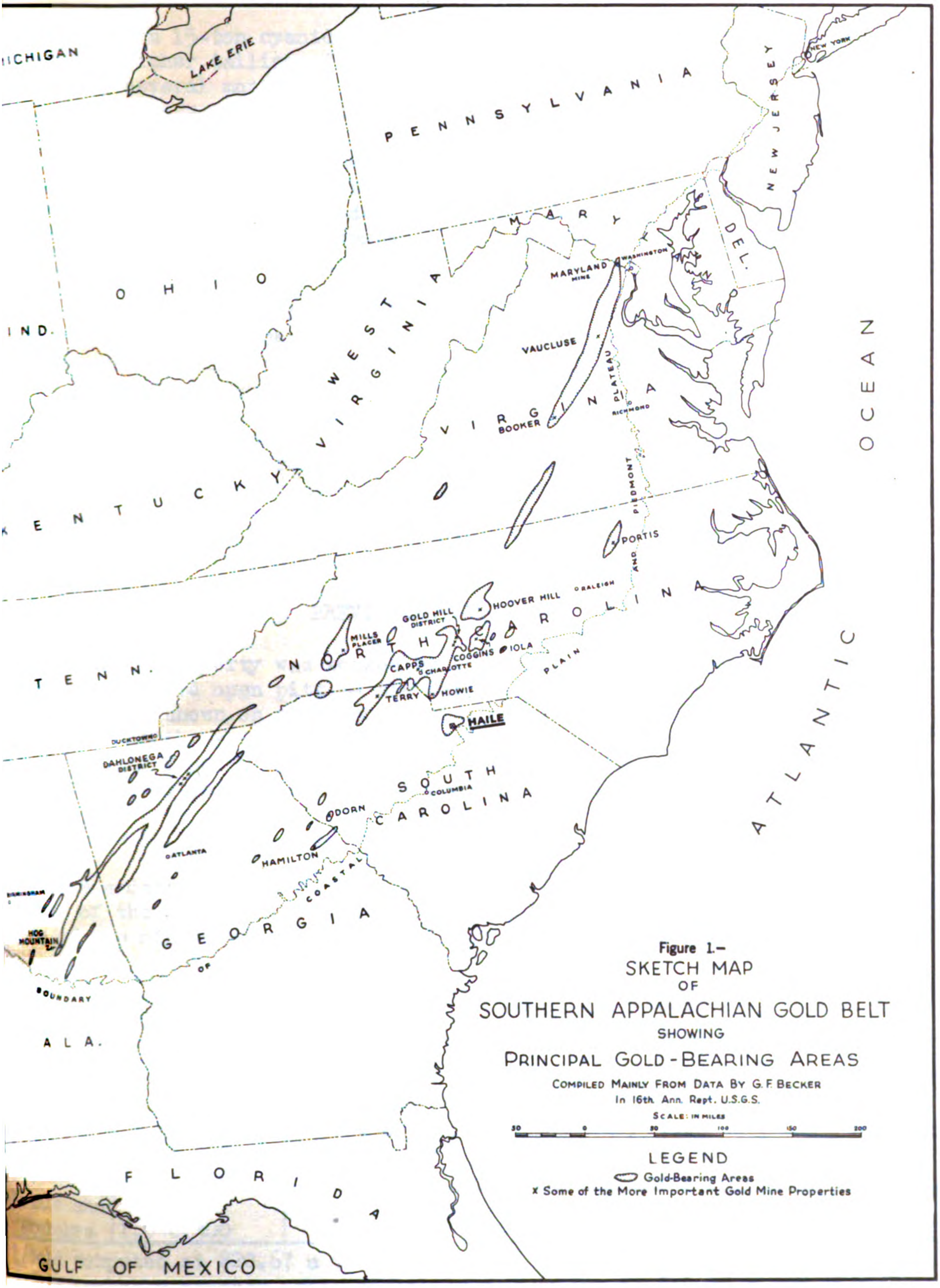

Figure 1.—
SKETCH MAP
OF
SOUTHERN APPALACHIAN GOLD BELT
SHOWING
PRINCIPAL GOLD-BEARING AREAS
COMPILED MAINLY FROM DATA BY G.F. BECKER
In 16th Ann. Rept. U.S.G.S.

In 1913, a 15-ton cyanide plant was erected on the property to treat some of the richer tailings from former operations. This mine was abandoned after several months of operating losses resulting from lack of competent preliminary metallurgical investigation.

In 1914, some pyrite was mined from one of the deposits on the property for use in the manufacture of sulfuric acid. In 1917 a mill was constructed to concentrate 300 tons of crude pyrite ore a day. This mill operated below capacity from March to November 1918, when, coincident with the close of the World War, the domestic market for pyrite collapsed.

In 1934, following increase in the price of gold, Haile Gold Mines, Inc., contracted to purchase the property. This operation was financed privately through the Bralowe Corporation, a mine-management and investment firm of New York City, of which the members are Harlan H. Bradt, W. Lunsford Long, and Hewitt S. West. In 1935 a 100-ton pilot plant was built near one of the old open pits, from which ore was being mined in 1908, when the former operations were suspended. From that time until April 1936 about 18,000 tons of ore was treated and 15,000 tons of tailings was impounded. The process used in this plant was simple amalgamation on plates, and only about 17 percent of the gold was recovered while nearly $5 per ton was left in the reject.

PREVIOUS DEVELOPMENT

The Haile property was worked by both open-cut and underground methods. There are five old open pits — the Haile, Bumalo, Blue Pool, Blauvelt, and Beguelin — as shown on the accompanying maps. Some of these pits were over 150 feet in depth, according to old records, and there were several vertical shafts ranging from 50 to 350 feet in depth. The deepest workings, near the Haile pit, are reported to have extended to a vertical depth of 475 feet.

PREVIOUS PRODUCTION

From a detailed study of the published literature and from company records of the pilot-plant operation, the total previous production and grade of ore mined during the several periods may be summarized as follows:

Period	Crude ore, tons	Value[1] per ton	Gross Value	Recovery percent	Net value produced
1829-77	133,000	$15.00	$2,000,000	50.0	$1,000,000
1878-80	15,000	11.00	165,000	60.0	99,000
1881-87	50,000	8.50	425,000	73.0	310,250
1888-1908	830,000	4.50	3,731,000	82.3	3,070,613
1935-36	18,300	3.88	71,000	17.6	12,320
Totals and averages	1,046,300	6.15[2]	6,392,000	70.2	4,492,183

[1] Gold computed at $20.67 a troy ounce.

[2] This probable average grade of ore is equivalent to $10.35 a ton at the present price of gold ($35 an ounce).

GEOLOGY OF ORE DEPOSITS5/

According to Schrader5/, unconsolidated, sandy, cretaceous sediments are underlain by pre-Cambrian schists and slates. The schists are cut by diabase dikes and by granite masses of carboniferous age. Three mineralized shear zones show silica and pyrite in quartz-sericite schists. There were two periods of pyrite mineralization in pre-Cambrian time -- one containing considerable gold and the other comparatively little. A general silicification accompanied the formation of the ore bodies with later quart veins and stringers. The quartz and pyrite do not always carry gold.

Weathering extends to a maximum depth of 150 feet. In this zone the schist has been altered to a soft material called saprolite, with an accompanying concentration of gold. In this oxidized zone, gold that originall intimately mechanically locked up in individual pyrite particles has been partly or wholly liberated by oxidation of the pyrite. Such gold can be recovered more or less effectively by amalgamation. Early operations, however, always met difficulties in an attempt to recover gold included in th unoxidized pyrite particles.

Mineralogically, average Haile ore as now mined consists essentially of quartz, kaolin, sericite, hematite, limonite, magnetite, gold-bearing pyrite, and some free gold. Occasionally a very little molybdenite, rutile copper and zinc sulfides, and arsenopyrite are found, but not enough of any of them to affect treatment of the ore by cyanidation.

Even in the oxidized zone pyrite is common, occurring as fine grains or crystals disseminated irregularly through the rock or concentrated locally in bands. Below the zone of oxidation, pyrite may range in amount from less than 1 percent to almost pure pyrite. Usually, 1 to 15 percent of pyrite is present in the ore now being mined.

At present ore is mined principally from the oxidized zone and averages about $5 in gold per ton (gold at $35 per ounce). However, the value are variable and may change from day to day within rather wide limits. Th ore is relatively soft and easily crushed, but some of the more highly sil cified and pyritized portions are tougher and more abrasive.

The normal specific gravity of the dry ore is approximately 2.6, but ranges from 2.2 to 3.2. Consequently, daily determinations are required to control the density of the mill pulps. The moisture content of the ore all of which is now mined by open-pit methods, varies somewhat with the weather but usually ranges from 5.5 to 10 percent.

5/Schrader, F. C., Pyrite at the Haile Mine, Kershaw, S. C., With a Note on Pyritization at the Brewer Mine Near Jefferson: In Geol. Survey Bull 725, 1922, pp. 331-345.

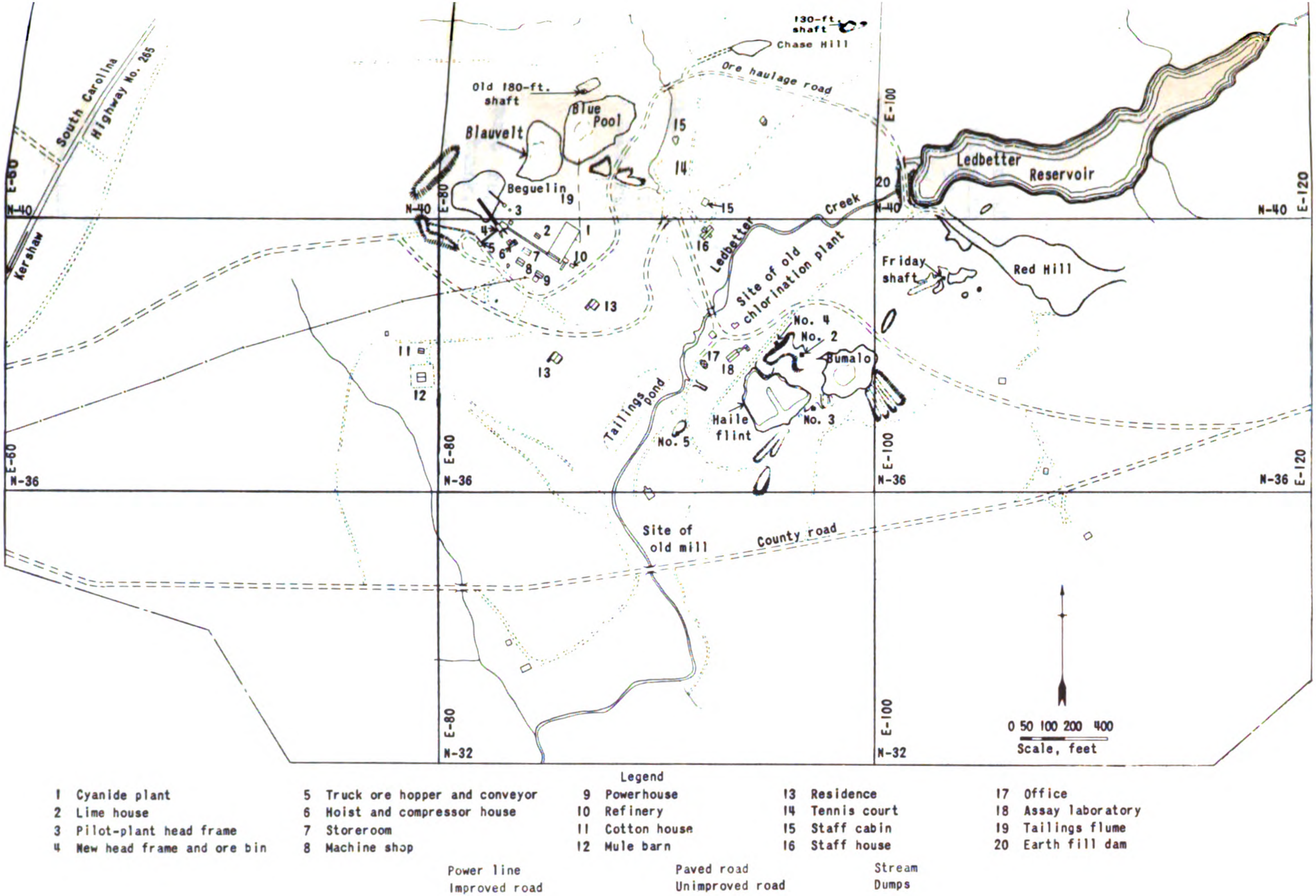

Figure 2.- Map of present producing area, Haile Gold Mines, Inc., Kershaw, S. C.

PRESENT OPERATIONS

Mining

Beguelin Pit

General

The situation of the Beguelin pit is shown in figure 2 on the map of the present producing area. Mining was under way in this pit when the boiler explosion in 1908 terminated operations. It was also from this pit that ore was mined for treatment by amalgamation in the pilot plant in 1935-36. The reject from this treatment was impounded on the hillside, above which the first unit of the cyanide plant was situated.

This unit of the plant was begun in June 1937. For the first 2 months it was fed solely with the pilot-plant rejects, which were sluiced to a sand pump for delivery to the mill.

In September 1937 mining was begun on a small scale in the Beguelin pit, and this newly mined ore supplemented the material from the pilot plant. For the next few months the tonnage of open-pit ore gradually increased while that of the sluiced rejects decreased, until by the end of February 1938 all rejects that could be sluiced economically had been handled.

Sluicing

Pilot-plant rejects were sluiced by water drawn from the main mill-supply storage tank. Pressure was boosted by means of an electric pump rated at 60 g.p.m. against a 200-foot head. Water was delivered through a 2 1/2-inch fire hose to a standard fire nozzle bushed down to give suitable velocity and volume. The rejects, which averaged about 80 percent through 200 mesh, sluiced readily.

The solids and water flowed to a head gate with removable boards and than by launder over a trash screen to a V-box, which fed a 2-inch Krogh sand pump driven by a 10-horsepower motor through a V-belt drive. The pulp, which varied greatly in density, was elevated 40 feet vertically and conveyed in a 3-inch pipe a distance of 300 feet horizontally. Under ordinary conditions, with only occasional assistance, one man on day shift and another on night shift ran the nozzle, the sand pump, and water-booster pump.

As this operation was temporary a rather heavy depreciation rate was used to write off the preliminary expenditures. The following represents costs on the first 8,632 tons of material sluiced:

Item	Cost per dry ton
Supervision	$0.0012
Labor	.0458
Power	.0621
Water supply	.0299
Repairs and maintenance	.0145
Depreciation	.0434
Total	.1969

Early Open-Pit Mining

During the early period of mining in the Beguelin pit, rather crude hand methods were used. Drilling was done with jumper drills, each operated by two men. After blasting, the ore was shoveled into wheelbarrows and dumped into a small skip, which was hoisted up the old skipway that had served the pilot plant. In the head frame it passed through a 7- by 10-inch Allis-Chalmers jaw crusher to a small dump truck, which transported it about 600 feet to a temporary hopper built alongside the first unit of the cyanide plant. From this small hopper a belt conveyor transported the ore about 20 feet to the feed box of a 6-foot by 22-inch Hardinge conical ball mill, where it was picked up by the scoop feeder. During this early phase of mining, the overburden was shoveled by hand from the face of the pit and removed in wheelbarrows.

Recent Mining Practice

Since April 1938 a new skipway, head frame, hoist, and 300-ton coarse-ore storage bin have been in use. The relative situations of these are shown in plan on figure 3 and in section on figure 4. Although the Beguelin ore is hoisted by inclined skipway and dumped direct into one compartment of the coarse-ore bin, the surface plant is so arranged that ore may be brought by truck from other parts of the property. The ore is dumped on a flat bar grizzly through which the oversize lumps are sledged by hand into a hopper below. It is then fed by an apron feeder to a 24-inch conveyor belt, which elevates it to a separate compartment of the coarse-ore bin.

When mining operations were begun in late August 1937 the Beguelin pit was about 250 feet long, 175 feet wide, and 90 feet deep. The long axis of the pit strikes approximately northeast and southwest. The footwall dips to the northwest at an average of about 60°, although this flattens toward the bottom of the pit. Mining has progressed against the northwest walls and also southwesterly along the axis of the pit. In addition, two 10-foot benches have been removed from the bottom of the pit Consequently, in June 1939 the pit was approximately 350 feet long, 200 feet wide, and 110 feet deep.

As the Beguelin ore body is relatively limited in life, little attemp has been made to install labor-saving devices for handling the ore, although compressed air is used for drilling.

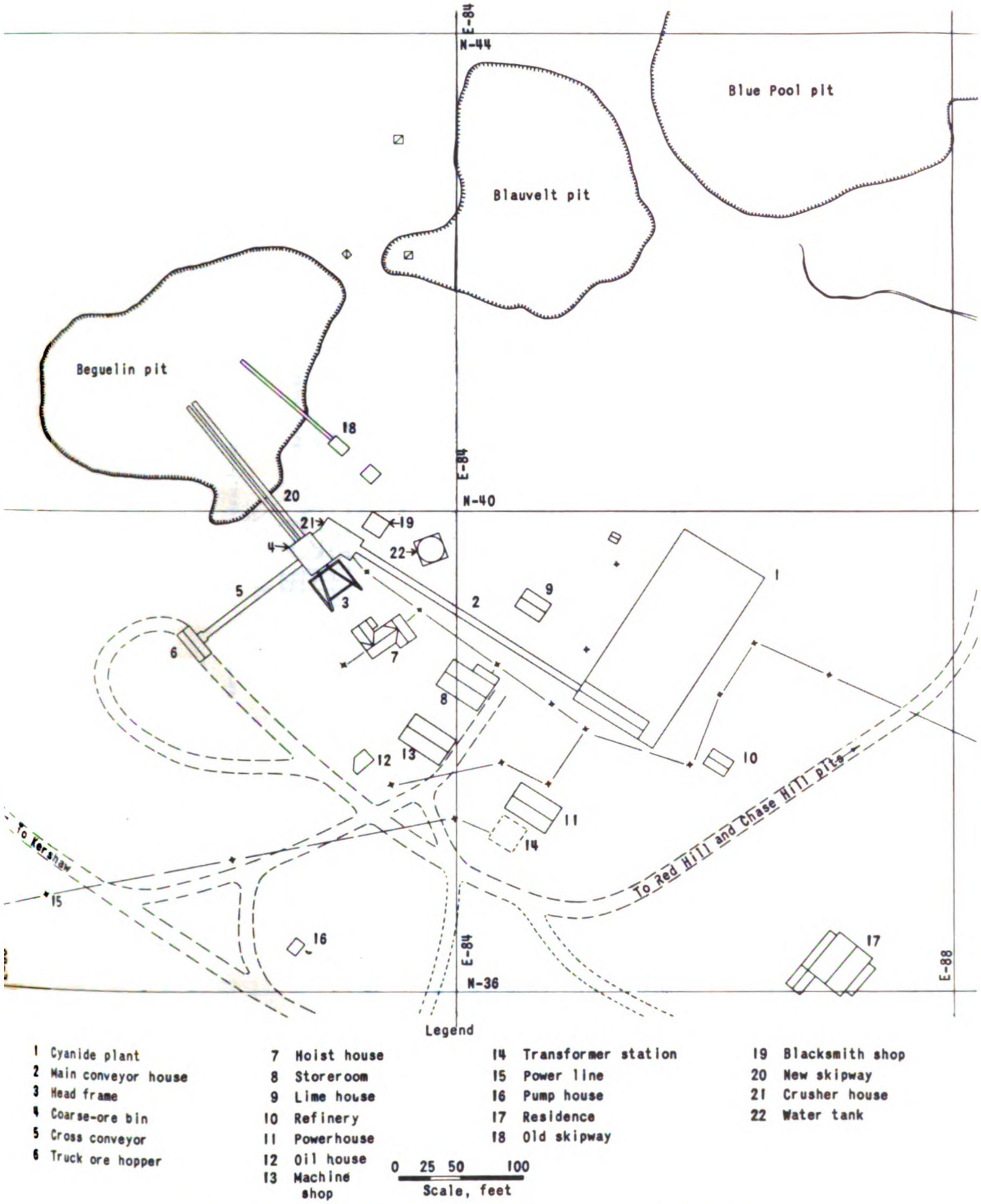

Figure 3.- Map of operations area showing cyanide plant and Beguelin pit.

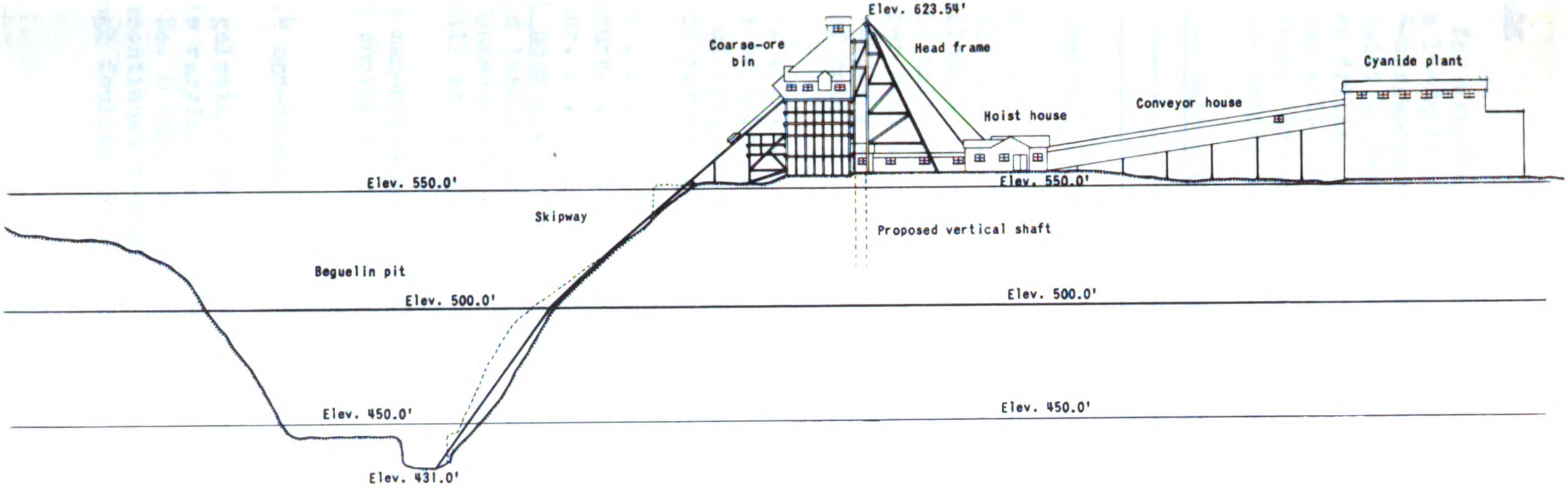

Figure 4.- Vertical section through coarse-ore bin, Haile Gold Mines, Inc.

Drilling and Blasting

Compressed air for drilling is supplied at 100 pounds gage pressure at the receiver by a 14- by 12-inch single-stage compressor driven through a flat belt by a 104-horsepower, 440-volt electric motor. All drilling is done on the day shift by two jackhammers (see table 1). Vertical holes about 8 feet deep are drilled in either the walls of the pit or in benching the bottom. Detachable bits are used entirely. These are sharpened by grinding in the shop. Experience shows that on an average these bits can be sharpened twice before they must be discarded.

TABLE 1. — Unit mining operations data, May 1939

		Beguelin	Red Hill
Tons:			
Total mined in month		6,152	4,524
Drilling:			
Total man-hours		570	990
Total holes drilled	number	541	775
Total feet drilled	do	4,195	5,812
Average depth of hole	feet	7.72	7.53
Depth drilled per hour	do	7.36	5.88
Rock per drill hour	tons	10.81	4.56
Rock per foot of hole	do	1.48	.78
Powder:			
Total powder	pounds	1,574	2,460
Powder per foot of hole	do	.376	.423
Powder per ton of ore	do	.256	.544
Ore per pound of powder	tons	3.92	1.84
Loading:			
Total man-hours		13,600	1,600
Total shovel-hours		—	245
Ore per man-hour	tons	.453	.453[1]
Ore per shovel hour	do	—	15.50

[1] This rate of hand loading for Red Hill is assumed to be the same as for the Beguelin. Consequently, the tons shoveled by hand and by power shovel for Red Hill are as follows:

	Tons
1,600 man-hours at 0.453 ton	720
245 shovel hours at 15.5 tons	3,804
Total tons	4,524

Therefore, 34.4 man-hours is equivalent to 1 shovel-hour.

Forty-percent gelatin dynamite is used for blasting the number of sticks used per hole varying with the burden. Electric firing is employed with instantaneous No. 6 caps on 10-foot iron lead-wires. Copper lead-wires have been discontinued because of their detrimental effect in the cyanide plant through fouling of solutions and precipitation on the zinc.

Loading

After shooting, the ore is sledged by hand into pieces weighing not over 25 pounds. It is then shoveled into rubber-tired wheelbarrows and wheeled to the loading platform, where it is dumped directly into the skip. The rubber-tired wheelbarrows have been found to be worth their extra first cost. They can be pushed over the relatively uneven pit floor more easily than the conventional type can be wheeled on planks.

Hoisting

The main skip used for hoisting ore has a rated capacity of 1-1/2 tons, although the average load, as shown by experience, is only about 1.2 tons. This skip runs on a skipway, which at the bottom of the pit has a 60° slope. It gradually flattens to a 40° slope, at which inclination it passes over the coarse-ore bin. The skipway is made of 18-foot sections, as shown on figure 5. These are bolted together and anchored to a concrete block at the surface. A general section of the pit, the skipway, coarse-ore bin, and head frame is shown on figure 4.

The 1 1/2-ton skip, which runs on 30-pound rails spiked to the skipway sections, has a 30-inch gage. The hoist has a single drum driven through gearing by a 40-horsepower, 3-phase, 60-cycle, 440-volt, 1,200 r.p.m., slip-ring, variable-speed motor. The rope speed is about 200 feet per minute. The hoisting rope is a 3/4-inch, 6 by 19, regular-lay, plow-steel cable. The drum carries 1-1/2 layers of cable. An electric signaling system connects the pit with the hoist room.

Hoisting is done on both day and night shifts. At night the pit is illuminated by 100-watt incandescent lamps on a 110-volt circuit and by a floodlight at the top of the head frame.

Drainage

Surface water is kept out of the pit as far as possible by means of trenches and flumes, which lead the run-off to one of the "branches". Mine water is removed from a sump under the skipway by electric pumps, which may be run separately or, in case of heavy rains, both may be used. At present a 4-inch pipe column takes the mine water to the surface, whence it is carried to one of the "branches".

Stripping

At first the overburden was stripped by shovel and wheelbarrow. Later, a second-hand Austin dragline with a 3/4-yard bucket and 40-foot boom on caterpillar mountings was used to remove the overburden from the rim of the pit. The Austin cast the material back, and as mining progressed to the southwest it had to be cast and recast several times. This machine was laid up a good deal of the time for repairs, and for this reason any attempt to give detailed information as to unit performance and costs would be misleading.

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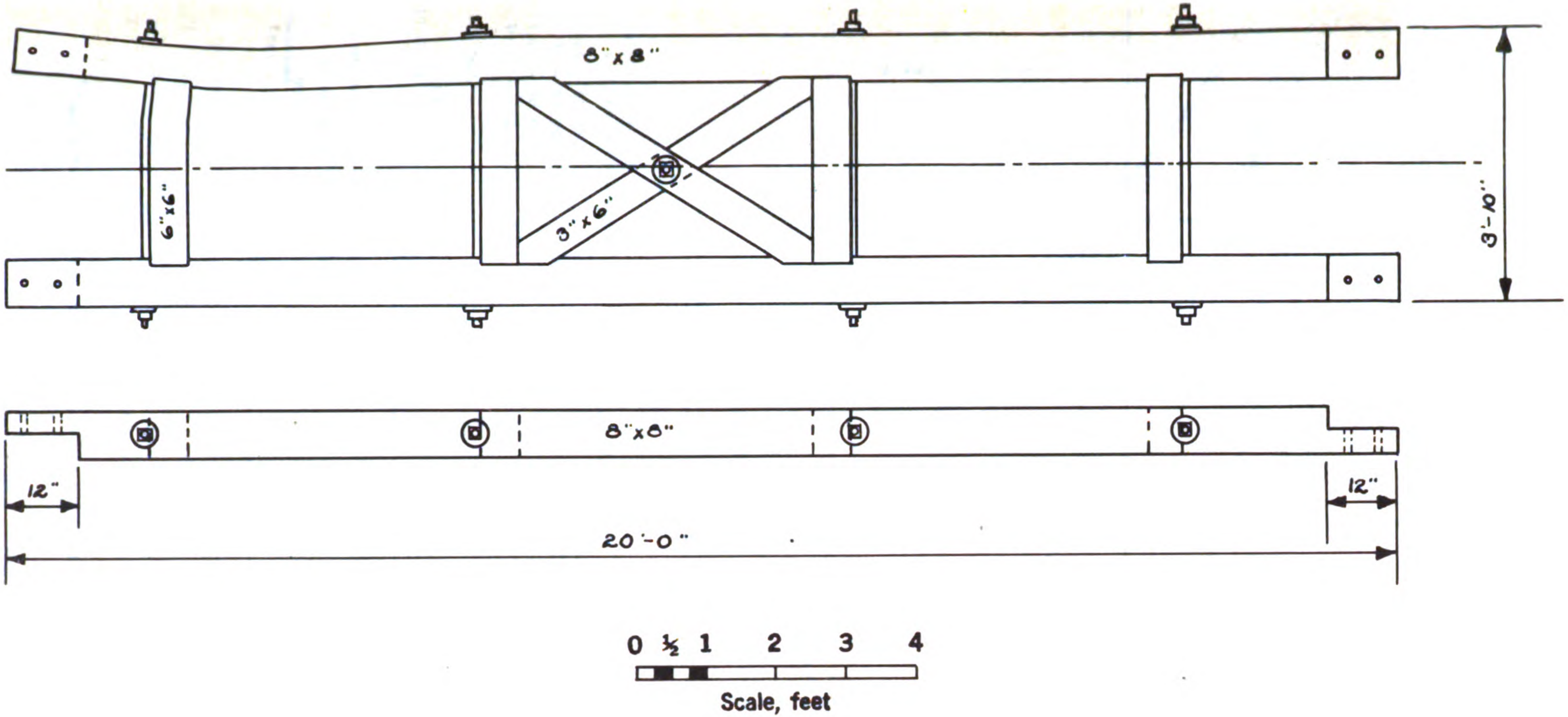
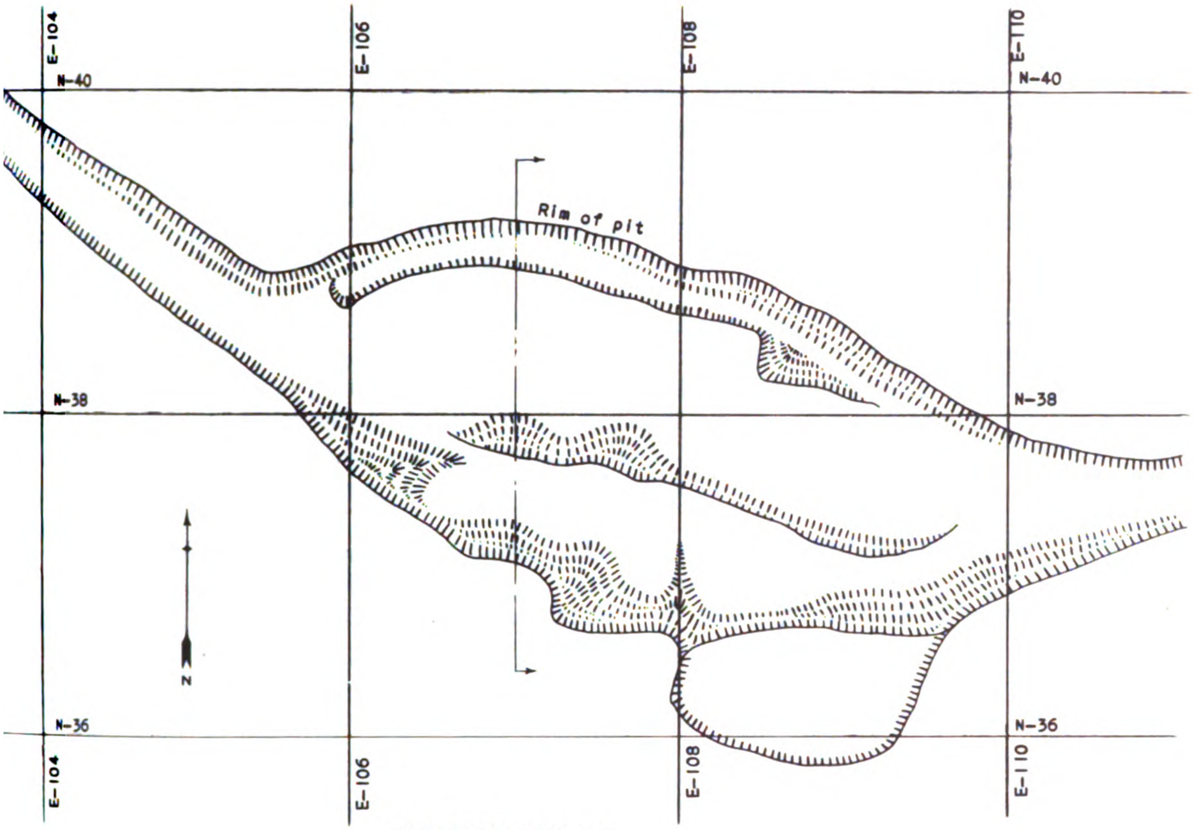
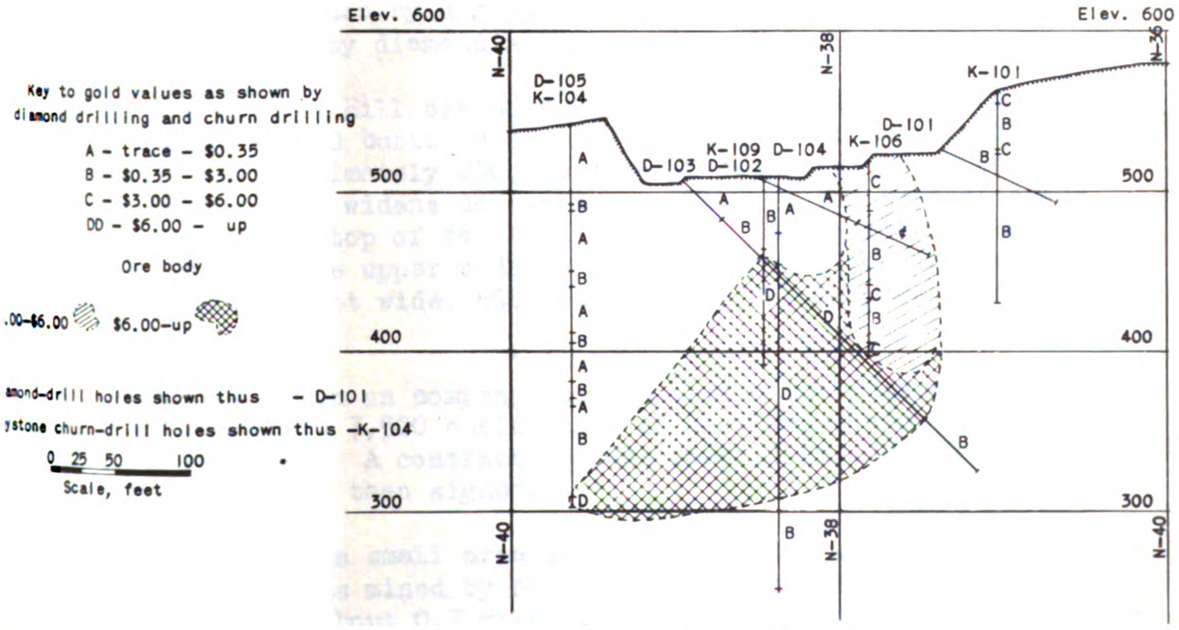


Figure 5.—Skipway sections for main hoisting skip, Haile Gold Mines, Inc., Kershaw, S. C.



PLAN OF RED HILL PIT



SECTION THROUGH PIT AND ORE BODY ON E-107 COORDINATE LINE

Figure 6.- Plan and section through Red Hill pit and ore body.

However, with a 180° swing, the Austin moved about 50 cubic yards an hour of actual working time. It was driven with a 65-horsepower gasoline-kerosene engine with hourly consumption of 3-2/3 gallons of fuel consisting of a mixture of one-half gas and one-half kerosene. The crew comprised one operator and one oiler. The weight of the machine was 60,000 pounds. The ropes used were discards from other operations. The 5/8-inch hoist rope was 120 feet in length while the 3/4-inch drag line was 60 feet. A clipped connection is used on one end on each of these ropes and a speltered socket connection on the other end.

The dragline was equipped with a Page bucket with four removable alloy-steel teeth. The price of the bucket was $600. A new bucket ran 1 year without repairs. The teeth, which were reforged locally, were sharpened after 1 to 3 weeks' service, depending on the material handled. It required 1 hour for two men to change a set of teeth. The purchase price of the machine when new was $8,000.

So much overburden has now been piled over the extension of the ore body to the southwest that it is uneconomic to strip further. The ore in this part of the ore body will be mined by underground methods.

Red Hill Pit

General

The situation of the Red Hill pit with respect to the rest of the operations is shown on the map of the present producing area (fig. 2). This ore body was outlined by a geophysical survey in November 1937 and was delineated further by diamond and churn drilling.

To prepare the Red Hill ore body for open-pit mining, it was estimated that about 130,000 cubic yards of stripping would have to be removed to uncover approximately 200,000 tons of open-pit ore. The ore body is lens-shaped and widens downward, as shown on the typical cross section (fig. 6). The top of the lens is only a few feet from the surface, but to uncover the upper side it is necessary to remove a wedge of overburden about 250 feet wide, 600 feet long at the surface, and 60 feet deep.

Stripping was begun on company account during the latter part of October 1938, and about 7,500 cubic yards of overburden was removed to the middle of December. A contract for the removal of 120,000 cubic yards of overburden was then signed.

By early December a small area of the top of the ore body was uncovered and some ore was mined by hand, loaded into trucks, and hauled to the coarse-ore bin about 0.7 mile distant. Later, a power shovel was purchased for loading ore into trucks, although occasionally some hand loading also was being done.

Drilling and Blasting

Compressed air for drilling is furnished by an electrically driven compressor at 100 pounds pressure.

Drilling is done at Red Hill on two shifts, a wagon drill with two supplementary jackhammers being used on each shift. Vertical holes about 8 feet deep are drilled dry. Detachable bits also are used at Red Hill.

Blasting practice corresponds to that at the Beguelin.

Loading

Some ore was loaded by hand in June 1939 at Red Hill, but most of the ore now is loaded by a 5/8-cubic-yard, heavy-duty, gasoline shovel. This machine is rugged and powerful for a 5/8-cubic-yard bucket. Working continuously, it could handle about 50 tons of ore per hour. The present rate of mining is only about 200 tons per day. When more ore is desired from Red Hill, this shovel can readily handle more tonnage.

This 5/8-cubic-yard shovel is also used to clean off the overburden at the ore contact and to deepen the approach to the pit out of which the ore is hauled.

The direct cost of operating the 5/8-yard shovel in the Red Hill pit in May 1939 was as follows:

TABLE 2. — Direct mining costs, May 1939

	Beguelin		Red Hill	
	Total	Cents per ton	Total	Cent per
Tons mined	6,152		4,524	
Drilling and blasting:				
Labor	$150.74		$180.28	
Supplies	481.85		452.79	
Compressed air	159.00		304.40	
Repairs	99.11		7.02	
Subtotal	890.70	14.48	944.49	20.
Loading:				
Hand:				
Labor	$2,148.77		$250.60	
Supplies	12.50		7.62	
Repairs	71.52		17.80	
Subtotal	2,232.79		276.02	

TABLE 2 - Direct mining costs, May 1939 (cont.)

	Beguelin		Red Hill	
	Total	Cents per ton	Total	Cents per ton
ovel:				
Labor....................	$0.00		$209.53	
Supplies.................	.00		75.55	
Repairs..................	.00		65.20	
Subtotal............	.00		350.28	
Subtotal, loading...	2,232.79	36.29	626.30	13.84
tal direct mining cost....	3,123.49	50.77	1,570.79	34.72
		Cost		
bor - 506 man-hours		$209.53		
pplies - Gasoline, 434.0 gals	$69.60			
Oil, 11.7 gals	5.95	75.55		
pairs		65.20		
Total direct cost		350.28		
urs operated - Total, 245				
Cost per hour		1.430		
ns mined - Total, 3,804				
Cost per ton		.092		

rect Mining Costs

Table 2 shows the itemized direct mining costs for both the Beguelin d Red Hill pits, which amount to $34.72 cents per ton of ore.

ansportion of Ore to Mill

Automobile trucks with 2-cubic-yard power dump bodies were used in ne 1939 to haul the ore from the Red Hill pit to the grizzly at the uck-ore dumping point, a distance of 0.7 mile. Upon reaching the izzly the ore is dumped, and the truck returns immediately to Red Hill r another load.

At the rate that Red Hill ore was mined in May 1939, three trucks re required to haul it. All were equipped with 2-cubic-yard power dump dies.

The operating data for these trucks for May 1939 were as follows:

Trucks operated.................	number	3
Total truck-shifts.............	do	88
Total loads of ore.............	do	1,440
Total ore hauled...............	tons	4,524
Trips per truck shift..........	number	16.40
Average truck load.............	tons	3.14
Truck shift....................	do	51.50

The direct cost of truck operation in May 1939 follows:

	Total	Per ton
Tons of ore hauled	4,524	
Labor	$189.16	$0.0418
Supplies - Gas 1,042 gal. $167.23		
Oil 37 gal. 19.48	186.71	0.0413
Repairs	72.71	0.0161
Total direct costs	448.08	0.0992
Cost per truck shift	5.10	

Metallurgy

General

Preliminary metallurgical tests on Beguelin ore were made during 1935 and 1936. Final tests made during the fall of 1936 indicated the desirability of using all-sliming cyanide treatment. Accordingly, the first unit of such a plant was designed for a capacity of 125 tons per day. Construction of this plant was begun on January 26, 1937, and feed was introduced into it on June 15, 1937; precipitation did not begin until July 1, 1937.

Early in the spring of 1938 it was decided to install the second unit of this plant. Experience had shown that the first unit could handle about 165 tons of ore a day. Therefore, a total tonnage of 350 tons a day was decided upon for both units. Virtually the only difference between the two units is that somewhat larger capacity was provided in the No. 2 unit for agitation and countercurrent decantation.

Operation of the No. 2 unit was begun November 22, 1938, and precipitation December 1, 1938.

Coarse Crushing

Beguelin ore is broken by sledges in the pit to a maximum dimension of about 6 inches. Red Hill ore is dumped from trucks on a horizontal grizzly, the bars of which are spaced about 5 inches. Coarse lumps are broken by hand sledging until they pass this grizzly and fall into the hopper below. From this hopper the ore is fed by a 24-inch by 7-foot apron feeder driven by a 2-horsepower motor to a 24-inch conveyor belt, which delivers it to a separate compartment of the coarse-ore bin. This conveyor belt, which has a total length of 120 feet, is inclined at an angle of 16° and is driven by a 2-horsepower motor.

As shown in the flow sheet (fig. 7), Beguelin ore is hoisted by skip and dumped automatically into one compartment of the coarse-ore bin. From this it is fed by a 30-inch by 5-foot apron feeder to a picking belt 30 inches wide by 15 feet between centers of head and tail pulleys. About all that can be removed on this picking belt is fragments of wood, tramp iron, and blasting wire.

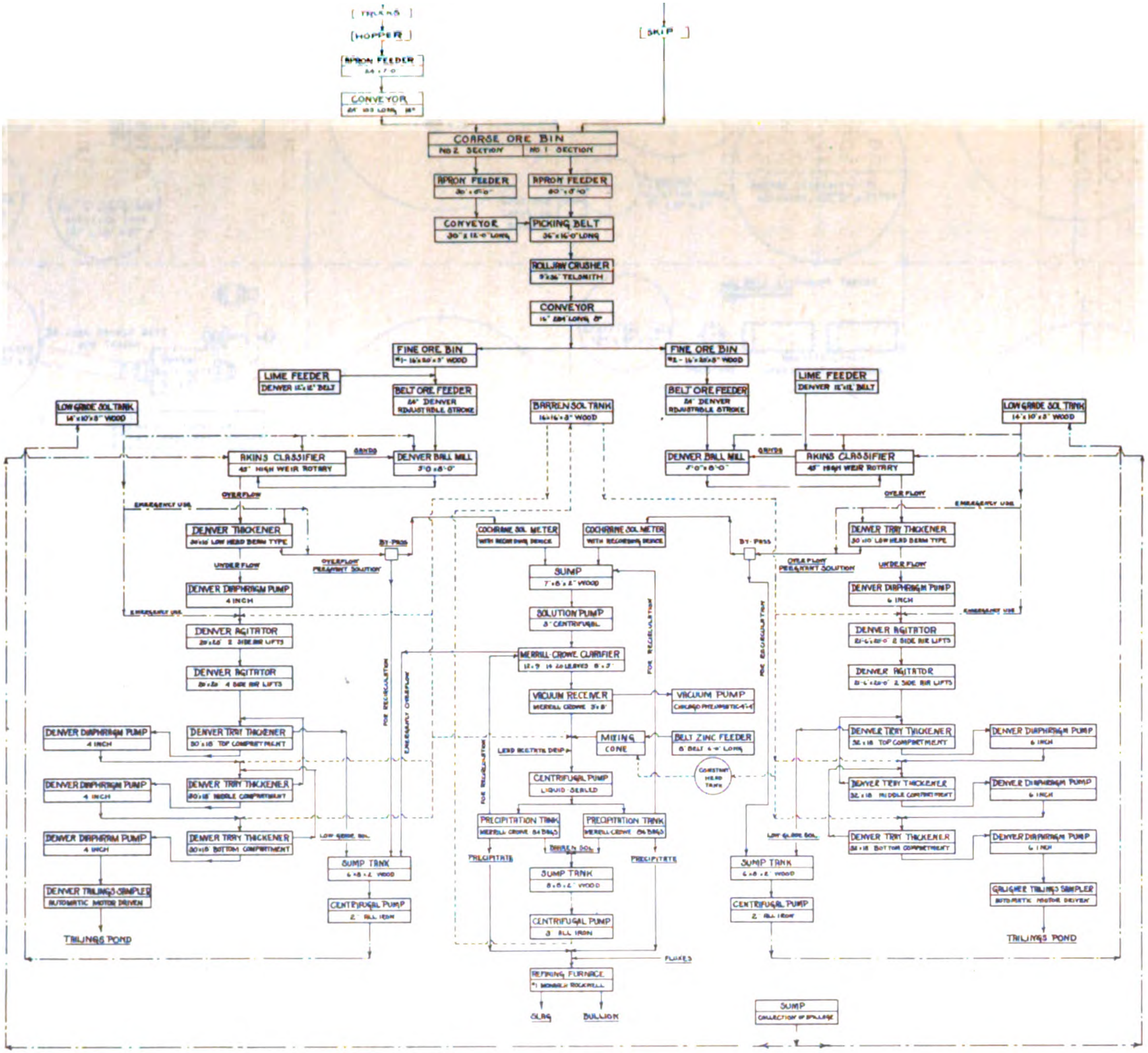


Figure 7.—Flow sheet, Nos. 1 and 2 units of the 350-ton cyanide plant, Haile Gold Mines, Inc., Kershaw, S. C.

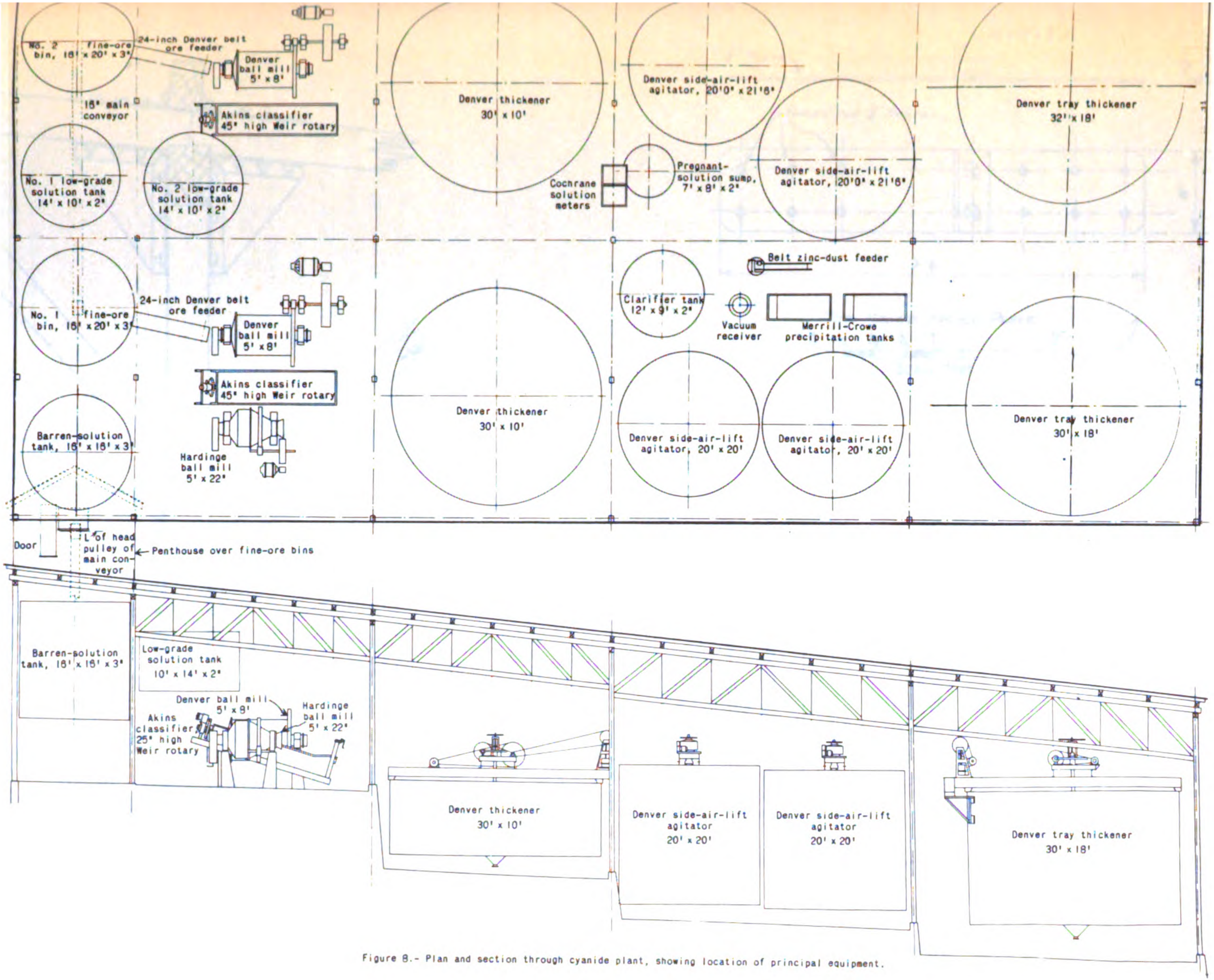

No. 2 fine-ore bin, 18' x 20' x 3'
24-inch Denver belt ore feeder
Denver ball mill 5' x 8'
18" main conveyor
Akins classifier 45" high Weir rotary
No. 1 low-grade solution tank 14' x 10' x 2"
No. 2 low-grade solution tank 14' x 10' x 2"
Denver thickener 30' x 10'
Denver side-air-lift agitator, 20'0" x 21'6"
Cochrane solution meters
Pregnant-solution sump, 7' x 8' x 2"
Denver side-air-lift agitator, 20'0" x 21'6"
Denver tray thickener 32' x 18'
No. 1 fine-ore bin, 18' x 20' x 3'
24-inch Denver belt ore feeder
Denver ball mill 5' x 8'
Akins classifier 45" high Weir rotary
Barren-solution tank, 18' x 18' x 3'
Hardinge ball mill 5' x 22"
Denver thickener 30' x 10'
Belt zinc-dust feeder
Clarifier tank 12' x 9' x 2"
Vacuum receiver
Merrill-Crowe precipitation tanks
Denver side-air-lift agitator, 20' x 20'
Denver side-air-lift agitator, 20' x 20'
Denver tray thickener 30' x 18'
Door
℄ of head pulley of main conveyor
Penthouse over fine-ore bins
Barren-solution tank, 18' x 18' x 3'
Low-grade solution tank 10' x 14' x 2"
Denver ball mill 5' x 8'
Hardinge ball mill 5' x 22"
Akins classifier 25" high Weir rotary
Denver thickener 30' x 10'
Denver side-air-lift agitator 20' x 20'
Denver side-air-lift agitator 20' x 20'
Denver tray thickener 30' x 18'

Figure 8.- Plan and section through cyanide plant, showing location of principal equipment.

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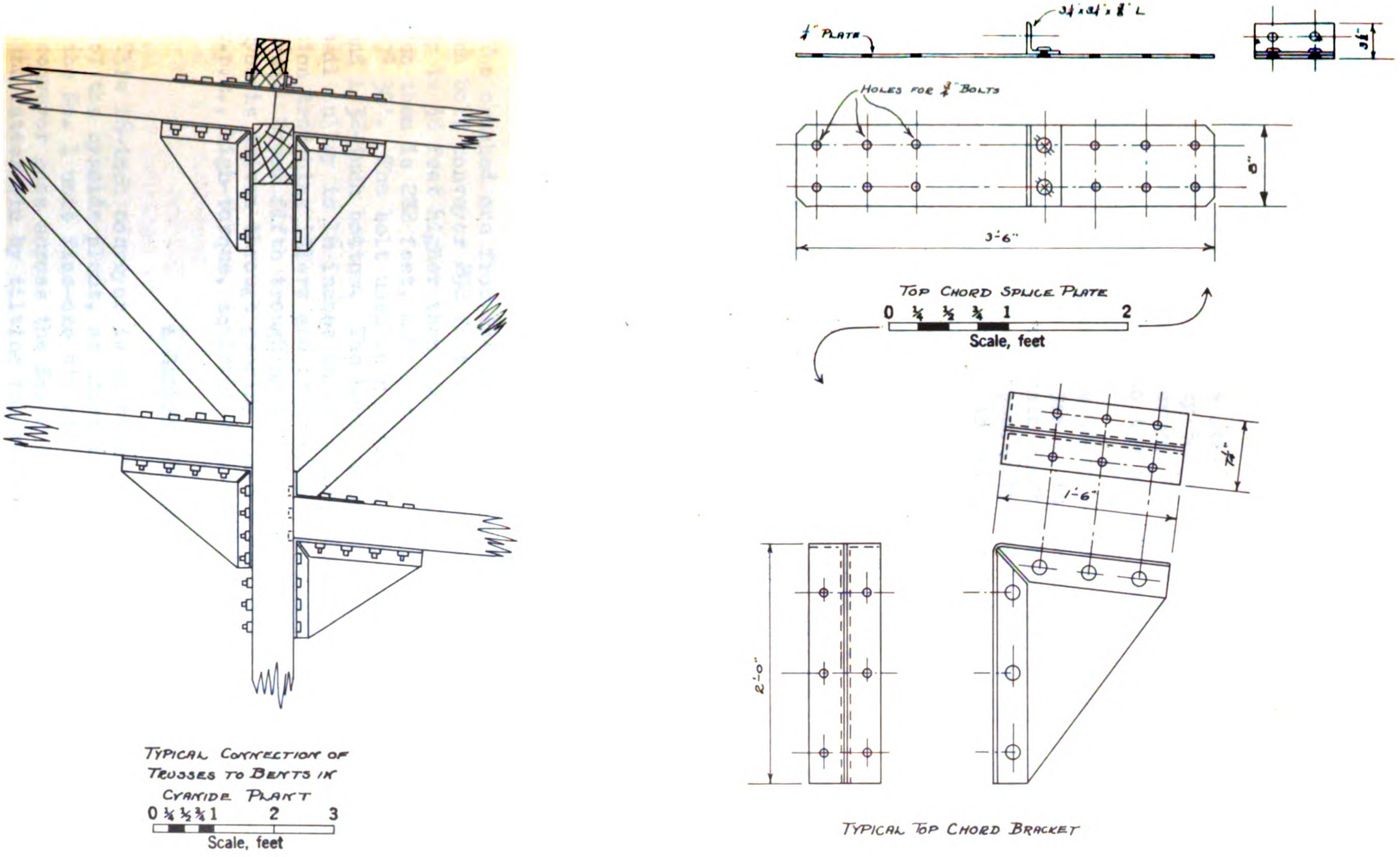

Figure 9.– Details of truss and bent connections and steel details in cyanide-plant building, Haile Gold Mines, Inc., Kershaw, S. C.

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It is not possible, with Haile ore, to identify waste rock visually and pick it out from the ore. The picking belt, however, serves effectively as a feeder to the jaw crusher.

Red Hill ore is fed from the other compartment of the coarse ore bin by means of another 30-inch by 5-foot apron feeder. This feeder dishcarges on a short 24-inch conveyor belt set at right angles to the feeder, which delivers the ore to the picking belt.

Coarse crushing is done in a 92- by 36-inch forced-feed, roller-bearing jaw crusher. The swing jaw is supported and actuated by means of an eccentric bearing on the main shaft, while the lower end of the jaw is held closely to a fixed opening by means of a toggle with adjustable wedge plates. This crusher is driven by V-belts from a 40-horsepower, 440-volt induction motor equipped with a starter of the autotransformer type, with overload and under-voltage release. It brings the run-of-mine ore down to 1 inch maximum at the rate of about 25 to 30 tons per hour.

Preliminary plans provided for secondary crushing. It was anticipated, however, that much of the oxidized ore would be "gummy" in wet weather, so that single-stage crushing was installed initially with the idea that a secondary crusher could be put in later if it were found to be necessary.

Although the size of the material discharged from the primary crusher may seem to be coarse for ball-mill feed, Haile ore grinds readily, down to a certain point. This feed furnishes tonnage enough to satisfy the balance of the circuit.

Transportation to the Cyanide Plant

The crushed ore from the primary jaw crusher drops directly to a 16-inch belt conveyor 292 feet long from head to tail pulley. The head pulley is 38 feet higher than the tail pulley and the horizontal distance between them is 282 feet, making the angle of inclination somewhat more than 7° 30'. The belt used is a 16-inch, 4-ply, 28-ounce duck, 1/8-inch top and 1/32-inch bottom. The head pulley is 24 inches in diameter, while the tail pulley is 16 inches in diameter. Standard 3-roll, 5-inch, antifriction troughing idlers and standard 5-inch antifriction return idlers are used. Every fifth troughing idler is of the self-aligning type. The conveyor is driven through gearing and a V-belt drive from a 3-horsepower, 860-r.p.m., high-torque, squirrel-cage motor.

Fine-Ore Storage

This 16-inch conveyor is at right angles to the long axis of each unit of the cyanide plant, as shown in figure 8. The head pulley is over the No. 1 unit fine-ore bin and the belt dishcarges directly into it. This conveyor goes across the No. 2 unit fine-ore bin and ore is discharged into the latter bin by tilting the belt with a home-made device that discharges the ore from the side of the belt.

Both fine-ore bins are circular tanks 16 feet in diameter and 20 feet in height. These are more heavily banded but otherwise are identical with solution tanks. A lining of steel plate is desirable near the front of these bins, near their discharge to the feeders. Each of these circular fine-ore bins has a capacity of 3,660 cubic feet. Allowing 20 cubic feet per ton of broken ore, this is equivalent to a storage capacity of 180 tons for each unit, or about 1 day's run of ore.

The hoppers of the belt feeders are set under the bottom of the bins near the front. The feeders are 24-inch by 15-foot adjustable-stroke belt ore feeders. They are set at an inclination and deliver the ore directly to the drum feeders of the ball mills of each unit.

To give access for barring down the ore in case of a hang-up over the feeder hoppers, a louver opening was designed by the mine staff, as shown by figure 10. At times the ore is damp and hangs up badly. Men are then sent into the bins to bar it down. Ropes with safety belts are provided for the safety of these men.

Cyanide Plant Flow Sheet

Fine Grinding

For the first few months, the feed to the No. 1 unit comprised tailings from the pilot plant. These already had been ground to 80 percent through 200 mesh, so that a 6-foot by 22-inch Hardinge mill that had been used in the pilot plant was the only grinding unit used for the first 8 months of operation. On March 1, 1938, a 6-foot by 22-inch used Hardinge mill was installed in parallel with the first mill but was soon replaced by a 5- by 8-foot cylindrical mill identical with that selected later for the No. 2 unit. Use of the 6-foot by 22-inch Hardinge mill was then discontinued temporarily.

The 5- by 8-foot cylindrical mills for each unit are equipped with combination drum and scoop feeders and are driven at 29 r.p.m. by V-belts from 100-horsepower, 865 r.p.m., 440-volt, slip-ring motors with drum-type starters.

Grinding is done in a low-grade cyanide solution. The density of the ball-mill discharge is maintained at about 85 percent solids by weight. Four-inch-diameter forged-steel grinding balls are used. About 200 pounds of balls a day is charged into each mill, and the ball consumption is about 1.15 pounds a ton of ore.

The mills originally were equipped with hard cast-iron liners. Although this material was fairly satisfactory for the end liners it was not suitable for shell liners, and the latter were replaced by liners of manganese steel. These are still in service, and, from inspection, it is anticipated that they should grind about 42,000 tons of ore. A set of shell liners weighs about 11,500 pounds.

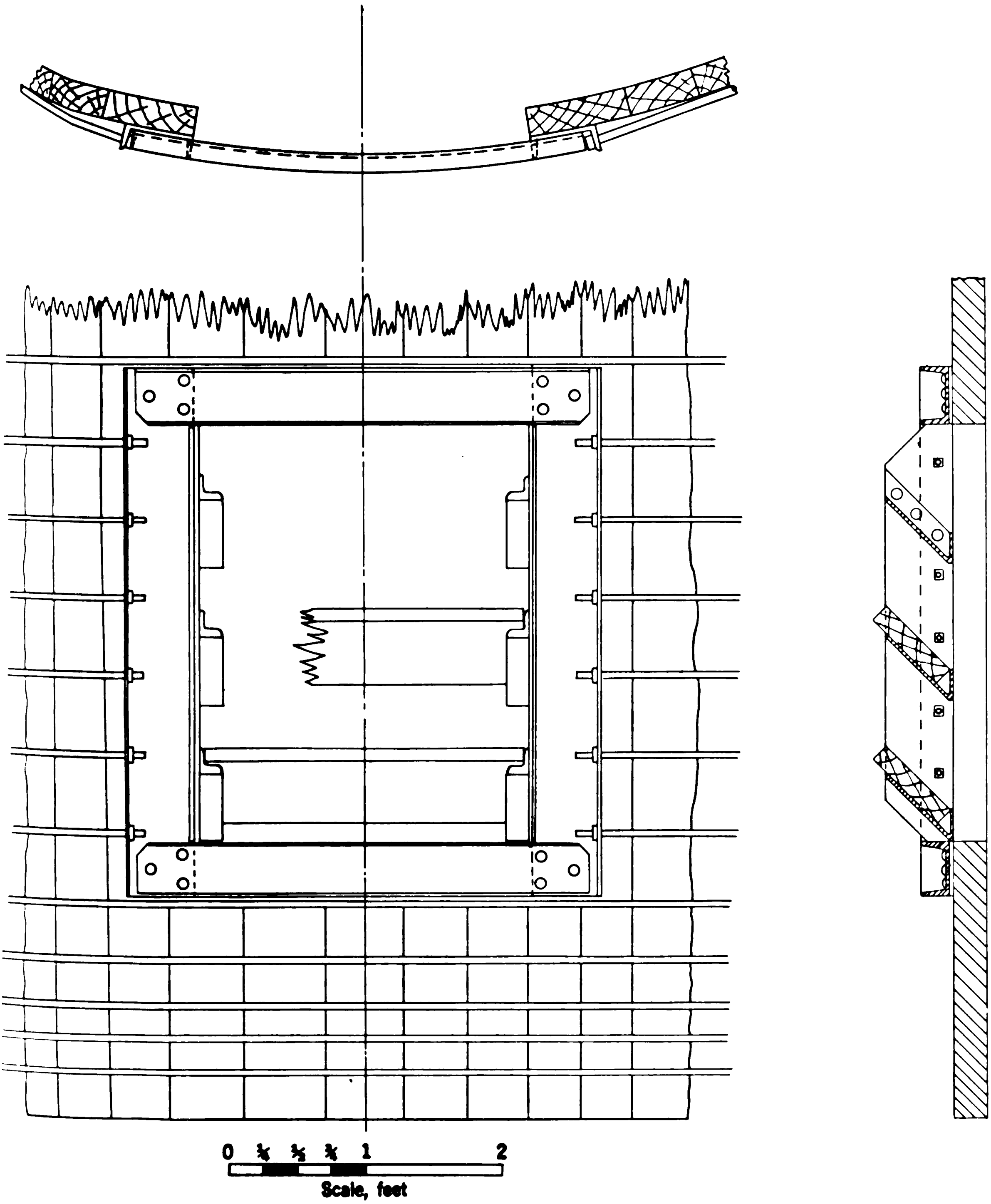

Figure 10.—Louver opening in fine-ore bins, Haile Gold Mines, Inc., Kershaw, S. C.

Scoop lips for the ball-mill feed scoop are made of cast iron and have a life of 7 to 9 days. A trial installation of manganese-steel lips gave virtually the same life but cost much more.

Classification

The ball mill of each unit operates in closed circuit with a 45-inch Akins high-weir classifier. In the No. 1 unit the classifier is set on a slope of 3-1/2 inches per foot, whereas in the No. 2 unit it is set at 3 inches per foot. The spiral on the No. 1 unit classifier revolves at a speed of 4-1/3 r.p.m. and that in the No. 2 unit operates at 5 r.p.m.

The overflow from the 45-inch Akins classifiers flows to the feed well of the primary thickener of each unit.

Primary Thickening

The amount of gold dissolved in the grinding circuits varies with the character of the ore; it ranges from 40 percent of the total amount of gold recovered when considerable pyrite is present to 70 percent at such times when virtually all the ore is oxidized material.

The primary thickeners, which receive the classifier overflow, are 30- by 10-foot Denver thickeners. The thickener mechanisms are mounted on wooden tanks, which reduces the volume somewhat. The rakes make one revolution in 5-1/3 minutes. The settling area of each thickener tank is about 685 square feet. These thickeners were selected when the No. 1 unit was being designed for a capacity of 125 tons a day, and at that rate there would be 5.48 square feet of settling area per ton per 24 hours. However, at present, by feeding each unit about 175 tons a day, this settling area has been reduced to only 3.9 square feet per daily ton. At the time the No. 2 unit was being designed, it was proposed to install a two-compartment primary thickener outside the building with a settling area equivalent to those now in use. It was then proposed that the two 30- by 10-foot thickeners be used for the No. 1 unit while the tray thickener would be used for the No. 2 unit. This was not approved, however, and the present settling area consequently is not large enough to settle the desired daily tonnage of certain of the oxidized ores from shallow depths, because they are very slimy and settle slowly.

The pulp is thickened to a density of about 60 to 62 percent solids by weight and is pumped to the agitators. In the No. 1 unit a 4-inch diaphragm pump is driven at a speed of 60 strokes per minute by the motor that drives the thickener arms. On the No. 2 unit a 6-inch diaphragm pump is driven at 16 strokes a minute by a 3-horsepower motor.

The thickened pulp delivered by the diaphragm pumps to the agitators is diluted at the pump discharge. Ordinarily, barren solution is used for this purpose, but in emergencies low-grade solution also is available at these points.

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Agitation

There are two Denver side air-lift agitators in each unit. The No. 1 unit agitators have 20- by 20-foot wooden tanks. The pulp is stirred mechanically and also is agitated by two air lifts in the first agitator and four in the second.

The No. 2 unit agitators consist of bolted-steel tanks 21 feet 6 inches in diameter by 20 feet in depth. Each is equipped with two air lifts. The agitator arms make about 5-3/4 r.p.m.

The pulp is agitated at a density of 49 to 50 percent solids by weight. At the present average rate of feed, the period of agitation is approximately 28.5 hours in the No. 1 unit and 34.4 hours in the No. 2 unit. During agitation, titration shows a decrease in strength of 0.05 pound NaCN and 0.025 to 0.05 pound CaO per ton of solution. Each agitator is operated by a 5-horsepower motor. Air is supplied by two single-stage rotary air compressors each rated at 135 cubic feet per minute at a pressure of 15 pounds per square inch. Each of these compressors is driven by a 10-horsepower motor.

Countercurrent Decantation

The overflow from the agitators goes to Denver tray thickeners with three compartments each. The No. 1 unit thickener is 30 feet in diameter by 18 feet in depth, while the No. 2 unit thickener is 32 feet in diamete by 18 feet in depth. Both thickeners are operated by identical 3-horsepower gear motors, and the rakes of each make one revolution in 10 minutes; they are equipped with overload alarms.

The No. 1 unit thickener is equipped with a 4-inch triplex diaphragm pump. This is operated at 56 strokes per minute by a 5-horsepower gear motor. The No. 2 unit thickener has a 6-inch triplex diaphragm pump operating at 18 strokes per minute and is driven by a 10-horsepower gear moto The stroke length of each pump can be adjusted while the pumps are operating.

Pulp is pumped at a density of 60 to 62 percent solids by these pumps. At these speeds and densities the 6-inch pumps with the slower speeds and usual shorter strokes have given almost no trouble from broken diaphragms, whereas the 4-inch pumps, driven at higher speeds and with a longer stroke, frequently have required new diaphragms.

The overflow from each tray thickener goes to its own solution sump, consisting of a 6-foot-diameter by 7-foot-height wood tank. From this it is pumped to the solution storage tank for the respective unit, at the head of the mill, by 2-inch centrifugal solution pumps. Each of these pumps is driven by a 5-horsepower motor through a V-belt drive.

The entire plant has sloping concrete floors in each bay. All pillage and floor washings flow down to a sump in the center of the mill n the lowest bay. From this sump it can be pumped either to the classiiers or to the No. 1 tray thickener.

recipitation

Solution overflowing from the primary thickeners goes to Cochrane olution meters, one for each unit. Those have a chart showing any variation in flow that can be integrated, but they also have a recording ntegrator that can be read directly. The capacity of each meter is 480 ons of solution in 24 hours for the particular size of V-notches used.

After passing through both meters, the solution flows to one sump, from which it is pumped by a 3-inch centrifugal solution pump to the larifier. This pump is direct-connected to a 2-horsepower motor.

The Merrill-Crowe "simultaneous clarification, deaeration, and precipitation" process is used at the Haile plant. The clarifier is a wooden tank 12 feet in diameter by 9 feet in depth, in which are suspended 20 5- by 8-foot Merrill-Crowe filter leaves.

The Crowe vacuum tank is 3 feet in diameter by 8 feet in height and is the standard type used in this process. The zinc feeder is the standard belt type, as are the mixing cone and the constant-head tank. All are made by the Merrill Co.

The vacuum for the vacuum tank is produced by a 4- by 4-inch vacuum pump. A 3-inch, water-sealed, centrifugal, solution pump driven by a 5-horsepower motor takes the solution from the vacuum tank and the zinc dust from the mixing cone and delivers it through a 2-1/2-inch pipe to the two precipitation tanks. Each of these precipitation tanks, which are 8 feet 8 inches long, 3 feet 4 inches wide and 3 feet 6 inches deep, contains 84 precipitate bags.

The barren solution flows to a sump tank 8 feet in diameter by 8 feet in height, from which it is pumped by a 3-inch centrifugal solution pump direct-connected to a 5-horsepower motor to a barren-solution storage tank 16 feet in diameter by 16 feet in height situated on the upper floor of the mill.

Plant Control and Practice

Sampling of Heads

Ore from the Beguelin pit is sampled after crushing by taking a cut across the 16-inch conveyor belt once each hour with a special flat scoop 3 inches wide.

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Ore from Red Hill, Chase Hill, or other parts of the property is sampled by taking a similar cut on the 24-inch conveyor belt before it is delivered to the coarse-ore bin. Two cuts are taken also from each truckload as it arrives at the horizontal grizzly. As the ore that has not been crushed is coarse, the results are not very precise but are accurate enough to indicate the approximate grade of ore coming from various parts of the property.

In the cyanide plant, separate samples are taken for each unit. Every hour a channel 3 inches wide is cut across the 20-inch feeder belts between the fine-ore bins and the ball mills.

Tonnage Determination

The tonnage of ore fed to each unit is estimated by weighing, once each hour, the quantity of ore on a length of belt equivalent to the diameter of the tail pulley. This is scraped off into a tub and weighed on ordinary platform scales. It is then shoveled back into the feed box of the ball mill. A "Productometer" revolution counter is attached to the shaft of the tail pulley of the feeder, and the number of revolutions per shift is recorded. A day sample and a night "grab" sample are taken from each belt for moisture determination. By averaging the weight of the samples taken during a shift after correcting for moisture and multiplying by the number of revolutions of the feeder tail pulley, the dry weight of ore fed to the ball mill for that shift is ascertained. The form used for recording the primary data for each unit and each shift, together with the calculations to give tonnage, is shown in table 3.

Sampling of Tailings

Final tailings discharged from each unit are sampled automatically. On the No. 1 unit an automatic sampler cuts the stream of tailings once in 5 minutes. On the No. 2 unit an automatic sampler cuts the stream of tailings every 10 minutes. Tailings samples are caught in enameled buckets, which contain a small quantity of permanganate solution to stop the reaction. In the assay laboratory these tailings samples are decanted and the solids are washed thoroughly. The solution and washings are measured by volume and the solids are dried and weighed. From the separate assays of the solids and the solution, the soluble loss and the gold content of the total tailings are computed.

Daily Value Precipitated

This is computed from the tonnage of solution precipitated and the assay of pregnant and barren solutions.

The tonnage of solution precipitated from each unit is obtained in the following manner: The integrated "units" of solution are recorded at the beginning and end of each shift for each meter. By means of a factor determined by occasional calibration, these units are converted into tons of solution.

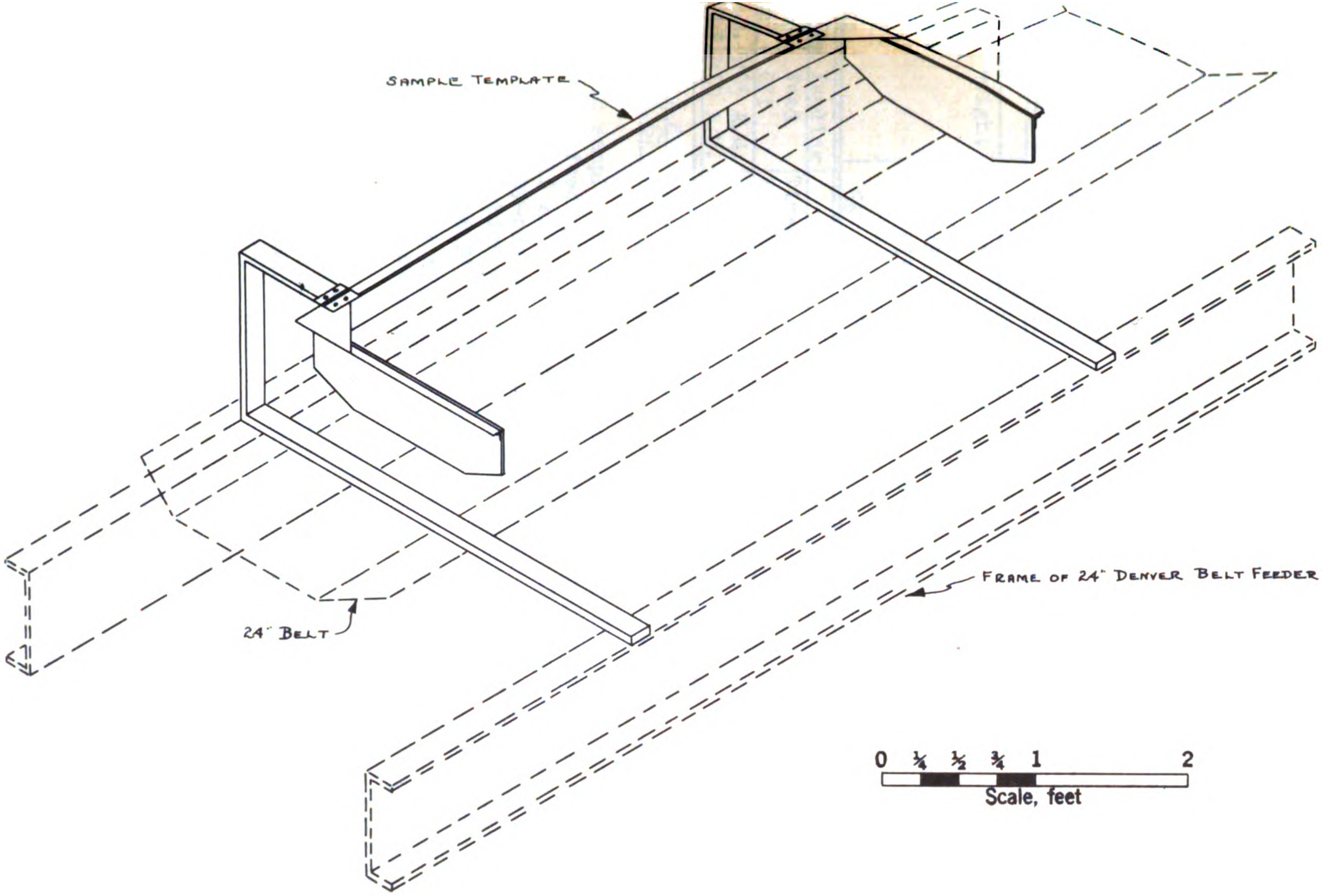

Figure 11.—Perspective sketch of sample template on 24-inch belt ore feeders to 5-by 8-foot Denver ball mills, Haile Gold Mines, Inc., Kershaw, S. C.

TABLE 3. — Tonnage report, unit No. 2, June 1939

ay shift		Night shift		Graveyard shift			Meter	
e	Weight	Time	Weight	Time	Weight	Time	reading	Revolutions
0	194	3:30	204	11:30	205	7a.m.	469,820	
0	196	4:30	206	12:30	197	3p.m.	470,525	706
0	198	5:30	197	1:30	189	11p.m.	471,186	660
0	201	6:30	193	2:30	192	7a.m.	471,782	596
0	203	7:30	190	3:30	201			
0	197	8:30	196	4:30	208			
0	204	9:30	204	5:30	194			
0	196	10:30	198	6:30	199			

	Day	Night	Graveyard	Total
ge net weight	198.63	198.50	198.13	
ent dry	93.5	96.4	96.4	
weight	185.72	191.35	191.00	
ev. ,000	.353	.330	.298	
tons	66	63	57	186

Time Feeding			Reason Off	Remarks
Feed On	Feed Off	Time Fed		
A.M.	5:05 A.M.	22:05	Feeder belt broke	
5 A.M.	7:00 A.M.	1:35		

Hours grinding	Percent possible time
23.67	98.63

A drip sample of pregnant solution is taken from each unit every shift from the pipe feeding each flowmeter. The drip pipe and cock is at an elevation about half an inch above the bottom of the V-notch. In this way, if the solution is by-passed or shut off from the flowmeter, the drip automatically stops, which precludes failure to shut it off or turn it on when stopping or starting precipitation.

A drip sample of the barren solution also is taken from each precipitation tank during every shift.

The estimated daily value of precipitate is computed by multiplying the tons of solution precipitated each shift by the net dollar value precipitated per ton. This latter figure is obtained by subtracting the assay value of the barren solution from the assay value of the pregnant solution.

The sum of the values precipitated for each unit during the three daily shifts is posted on the metallurgical control sheet as the estimated dollar value of the precipitate for that day. At every clean-up the estimated value of the precipitate is checked against the weight and assay of the bullion produced.

Metallurgical Control Sheet

Wide columnar forms are used for this purpose. In the upper part of the sheet are the daily data for the No. I unit, in the middle are the data for the No. 2 unit, while at the bottom are the totals for both units One such sheet serves for 10 days and three sheets are used each month.

The daily ore tonnage is posted in its proper column. Also, tons of solution, assay of pregnant solution, assay of barren solution, the calculated value of precipitate, and the assay of tailings are inserted in thei respective columns.

From the tonnage of ore and the assay of tailings the dollar value i the tailings is computed. This plus "dollars in precipitate" represent total "dollars accounted for." This total is then carried to the column for "dollars in feed," and then, by dividing this by the "tons of ore fed" the calculated grade of feed is obtained. Another column shows the assay of the direct sample of feed for comparison.

Daily Report Sheet

From the metallurgical control sheet a daily report is made and sent to the New York office, the form of which is shown in table 4.

At the bottom of this sheet the essential results for the day are compared with the weighted average results from the beginning of the month and also with a 30-day "moving average." This represents the weighted average for the previous 30 days. It will be noted on the example given that on the 30th day of each month the monthly average is identical with the 30-day moving average.

Routine Mill Tests

Pulp density. — Hourly density determinations are made of the pulp at the classifier overflow, primary thickener discharge pump, in the agitators, and on the underflow pumps from each of the trays in the tray thickeners.

The ball-mill discharge is maintained at about 85 percent solids, but this is not checked regularly. Classifier overflow was formerly 18 to 20 percent solids, but now it is 20 to 22 percent solids. All thickener underflows are 60 to 62 percent solids and they seldom deviate more than 1 or 2 percent from these figures. If they are permitted to get much thicker than this there is a danger of sticking the tray thickeners or of bursting pump diaphragms. This is particularly true if the specifi gravity of the crude ore should change.

All density determinations are made on the assumption that the dry o had a specific gravity of 2.6. Actually, the specific gravity ranges fro about 2.4 to 3.2, and is usually 2.5 to 2.7. On the above assumption, it is safe to have 60 to 62 percent solids with the above equipment.

TABLE 4. — Daily report, June 30, 1939
(From metallurgical control sheet)

	Unit 1	Unit 2	Total
Total tonnage	181	211	392
Assay per ton	$5.09	$4.37	$4.70
rinding - Hours	24.0	23.75	47.75
Percent possible time	100	99	99
atures - Mill solutions, degreesF.	94	93	—
tailings - Assay	$0.625	$0.521	$0.585
itate:			
gnant solution, assay	$1.505	$1.645	—
ren solution, assay	.003	.003	
ecipitated per ton solution	1.502	1.642	$1.612
s solution precipitated	533.50	494.46	1,027.96
llars precipitated, this day	$801	$812	$1613
llars precipitated, this month	$23,006	$20,339	$43,335
ecipitate on hand, first of month			15,744
Total			59,079
ansferred to "clean-up"			53,466
Precipitate now in bags, both tanks			5,613
ery - Accounted for, percent	88	88	88

ges - total production

	This day	Average this month	Moving average, 30 - day
ed - Total - tons	392	358	358
- assay	$4.70	$4.75	$4.75
ilings - Assay	$0.585	$0.716	$0.716
ecipitate	$1,613	$1,445	$1,445
traction-Calculated percentage	88	85	85

Signed________________________
General Manager

Strength of solutions. — Samples of each primary-thickener overflow nd barren solution are titrated for lime and free cyanide every 2 hours. olution strength is maintained at about 0.6 pound of free cyanide per ton f solution. It is somewhat lower than this in the tray thickeners, owing to the water added to the lowest tray.

When titration shows the solution to be weak in cyanide, it is brought up to strength by adding cyanide in the overflow launder of the tray thickener. This is the solution that is pumped to the low-grade solution storage tank principally for use in the ball-mill classifier circuit. It is added at this point for convenience, as no difference in extraction was noted when it was added in the agitators or at other points.

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Slaked lime is purchased in 50-pound bags and is stored in a separate lime-storage building. This is conveniently near the grinding floor of the mill. It is sheathed with matched boards inside the studs and is floored with concrete. The lime is wheeled into the mill in a rubber-tired wheelbarrow as needed. It is fed in the desired amount to the ball-mill classifier circuit of each unit by means of belt-type dry reagent feeders.

Barren solutions are tested for gold twice during each shift by Dowsett's method.

Mill Shift Report

Data on pulp density, solution strength, temperatures of solutions, and other matters are recorded for each unit and each shift on mimeographed forms, as shown in table 5.

Points Where Solutions Are Added

"Low-grade" solution, which overflows from the top tray of the tray thickener, is added to the pulp at the ball-mill-feed and ball-mill-discharge launders. The piping is so arranged that this solution can also be used in emergencies in the agitators or can be sent to the precipitation unit by way of the primary-thickener overflow launders.

Barren solution is used to dilute the agitator feed to about 50 percent solids by weight. It is also added to the pulp fed to the lowest tray of the tray thickeners. It can, moreover, be added to the middle tray of these thickeners, but this is not usually done. Barren solution is also added to the mixing cone of the zinc feeder.

Enough water is added to the feed of the lowest tray of the tray thickeners to replace the solution lost to the final tailings and by evaporation.

Treatment Time in Circuit

When 360 tons of ore a day is treated on normal pulp densities, the duration of the several stages of the process is as follows:

	Hours	
	No. 1 unit	No. 2 unit
Primary thickeners	5.1	5.1
Agitators	28.5	34.4
Tray thickeners	5.5	6.0
Total.	39.1	45.5

TABLE 5. — Mill-shift report

UNIT NO. 1

6-28-39

ents

charged - Lb. 300

ide - Cans opened - None

dust - Lb. to feeder - 9 at 8:45 a.m.

salt - Oz. to feeder - 9

s charged - Lb. - 70, 4-inch

SHIFT Night

SOLUTION METER READING

End of this shift - 46,500

End of previous shift 46,040

Difference 460

Tons solution precipitated - 126.8

PULP DENSITIES (percent solids at 2.6 spec. grav.) and TEMPERATURES

Primary thickener overflow, °F.	Primary thickener underflow		First agitator		Second agitator		Classifier overflow, Percent	Tray-thickener pump discharge			
	°F.	Percent	°F.	Percent	°F.	Percent		First		Second, percent	Tailings, percent
								°F.	Percent		
84	84	60	82	49	79	49	22	86	62	63	62
84	84	60	—	49	—	49	20	85	65	63	61
85	84	61	82	49	79	49	21	86	64	65	60
85	85	63	82	49	79	49	21	86	63	65	61
85	85	64	—	49	—	49	20	86	64	64	66
85	85	63	82	49	79	49	21	86	63	62	66
86	85	64	82	49	79	49	21	—	64	61	63
—	—	64	—	48	—	49	20	85	62	63	62

SOLUTION STRENGTHS

Tray thickener overflow			Pregnant solution		Barren solution			Tailings solution	
Degrees F	NaCN percent	CaO, percent	NaCN, percent	CaO, percent	NaCN, percent	CaO, percent	Au	NaCN	CaO
85	0.55	0.15	0.65	0.25	0.65	0.25	Trace, 4 p.m. Little more than trace, 9 p.m.		
85	.5	.125	.6	.25	.65	.25			
86	.5	.15	.6	.3	.6	.35			
86	.55	.15	.55	.3	.6	.35			

ed on: 7:35 p.m.	Feed off: 7:10 a.m.	Cause of delay: Tighten leaking liner bolt in ball mill and put on new scoop lip

rs: Repacked low-grade solution pump on this unit.

ks: Turned water on tray-thickener underflow pipe at 8:40 p.m.; off again at 10:00 p.m.

Signed________________________

Distribution of Values in Tailings

The total tailings average a loss of gold in solution equivalent to about 8 cents per ton of solids. The following screen analysis is fairly representative of the fineness of grinding and the distribution of gold and sulfur values in the tailings. As almost all the sulfur is present as pyrite, it is evident that the highest gold value is in the plus 325-mesh material because the greatest amount of pyrite is in the material of this size.

Size, mesh	Percent by weight	Gold		Sulfur	
		Dollars per ton	Percent by weight	Assay, percent	Percent by weight
-200	15.2	$0.70	16.42	3.51	10.33
+325	19.0	1.05	30.78	8.35	30.71
-325	65.8	.52	52.80	4.63	58.96
Total	100.0	.648	100.00	5.17	100.00

Slime Control

Much of the oxidized ore from shallow depths at the Haile produces a large amount of slime on grinding. This settles slowly, and many coagulating agents have been tried in attempts to accelerate the settling of such material, but nothing so far has proved as beneficial as heat.

As the daily capacity of the plant depends entirely on the settling rate of the material treated and as it was found from many laboratory tests that this rate was accelerated in warm solutions, two old sawmill boilers were obtained locally for use in heating the solutions. When the slime is bad, steam at low pressure is introduced into the thickeners. The steam pipe enters at the periphery of the tank and discharges below the feed well. In this way it is possible to raise the temperature of solutions as much as 30° F. and as a result of the accelerated settling rate at that temperature 50 to 100 tons a day additional may be treated satisfactorily.

When slimes are troublesome, the amount of lime is maintained at 0.2 to 0.3 pound per ton of solution. With a lime content as high as 2 to 4 pounds per ton of solution, a clearer overflow solution is obtained and the precipitation may be better, but the "curd" or floccules are very light and require a much longer time to settle to the required density. On the other hand, a lower lime content tends toward a denser, thicker underflow and permits more rapid settling, which means that a larger tonnage can be handled.

Precipitation

After the pregnant solution that overflows from each primary thickener is metered and sampled, it flows by gravity to a common sump, from which it is pumped into the clarifier.

While there is space in the clarifier for 20 filter leaves, custom- ily only 14 or 15 are used at one time. Each day two or three of these aves are washed to remove the cake of slime adhering to the outer sur- ace. The framework of the filter leaves is covered with burlap. Out- ide of this is a bag of Pequot sheeting, which is slipped over the burlap- overed frame and is then caulked at the top with rope. These outer bags sually have a life of about 6 months.

After being washed and before being returned to the clarifier, each ilter leaf is coated with a wash of Hi-Flow Supercel, a diatomaceous arth preparation, which is applied with a "whitewash" brush. This helps ery materially to obtain clear solutions quickly. To exclude air, the ilter leaves are entirely submerged in the clarifier.

A vacuum of 29 to 30 inches of mercury is maintained in the Crowe ank by means of the dry vacuum pump, and the oxygen is removed from the olution almost completely. Such complete deaeration of the clarified olution decreases the consumption of zinc dust and also prevents the ormation of white precipitate.

The belt of the zinc feeder travels 0.7 foot per hour. Zinc dust is eighed and spread evenly over a 5-foot length of the bolt. The usual mount is 9 pounds per 5 feet of belt, although this is always increased to 10.5 to 11 pounds when operation is resumed with new precipitate bags fter a clean-up.

The consumption of zinc dust ranges from about 0.031 to 0.037 pound er ton of solution precipitated. This is equivalent to 0.73 to 0.95 ound avoirdupois of zinc per troy ounce of gold. No allowance has been ade in this figure for the silver that is also precipitated.

The zinc dust falls into a mixing cone, and a liquid reagent feeder pplies a proper, uniform amount of lead acetato solution into which a strong cyanide solution sometimes is allowed to drip.

When the mixing-cone valve opens, the lead-zinc emulsion flows into he pipe carrying the deaerated solution from the vacuum tank and mixes ith it. The liquid-sealed pump conveys the mixture to the bags in the recipitate tanks. The flow from this pump is divided as evenly as ossible by valves between the two identical precipitation tanks, each f which contains 84 bags.

As the pregnant solution from both units is combined before clari- fication, deaeration and precipitation, each of these tanks receives aterial presumably identical in character. However, it is impossible to divide the precipitate and barren solution accurately between them, so that the bags are changed in both tanks at every clean-up. In this way, the estimated value of the precipitate for the clean-up period can be com- pared accurately with the bullion produced.

Clean-Up Practice

When cleaning up, the solution is drained in the clarifier until the tops of the filter leaves are exposed. The vacuum pump is shut down, the valves between the vacuum tank and precipitate pump are closed, and the zinc feeder is stopped.

The barren solution in the precipitation tank is drained to the barren-solution sump. The valve above the manifold is closed, and a compressed-air hose is connected to the manifold. Compressed air, at the same pressure used in the agitators, is then blown through the bags from 10 to 15 minutes to remove all solution possible.

The bag heads and attached bags are unscrewed from the manifold and allowed to drain. A new set of bags, previously attached to a spare set of heads, is then screwed onto the manifold, a very small amount of vaseline being spread on the threads to facilitate removal at the time of the next clean-up. Three manifolds for six bags each usually are removed for cleaning and are replaced with spare cleaned manifolds. These are changed in rotation. It requires approximately 1 hour for 4 men to change the 84 bags in each precipitation tank.

The drain valve on the precipitation tank is then closed and the valve opened to permit return of the solution to the clarifier. The solution pump is then started, and as soon as the bags appear to be filled with solution and free of atmospheric air a charge of 8 pounds of Hi-Flow Supercel is mixed with barren solution and gradually fed to the bags through the mixing cone. Considerable care is exercised to see that all bags get a proportionate amount of this charge. To accomplish this, the manifold nearest the center of the tank is kept closed to insure that the bags toward the outside edge of the tank receive their share.

A charge of 8 pounds of zinc dust is treated with about 3/4 pound lead acetate or nitrate and is then mixed with barren solution and slowly fed to the mixing cone. The zinc feeder is then started and the solution is allowed to recirculate to the clarifier for 30 minutes to an hour, or until the barren solution shows only a trace of gold by the Dowsett method. After this, the solution is sent to the barren-solution sump for circulation in the regular cycle.

The rate of feeding zinc dust is increased 30 percent for the first shift after clean-up and is reduced gradually to normal after a few days.

After the bags containing precipitate have drained for a time they are removed from the heads, and the outer canvas bags are stripped off to be used again. The precipitate is then dried in the inner bags for about 12 hours on a home-made drier located on the lower floor of the building. When the bags are dry, they are piled into a galvanized-iron tub and carried to the refinery.

Refining

Bullion is refined in a detached wooden building covered with corru-ated iron. Its concrete floor is 16 feet wide by 24 feet long. A ouver ventilator in the roof carries off the hot gases from the furnace.

The principal equipment in the refinery building is a metal-topped ork table and the melting furnace. The work table has a 4- by 8-foot op. Around the back and sides is a strip of heavy sheet metal 18 inches igh, which is welded to the table top. The top consists of a 1/2-inch late, which is heavy enough to permit dumping of bullion molds or pounding of slag and bullion bars on it without denting.

The melting furnace is a Monarch-Rockwell tilting, reverberatory-ype, D.C. furnace having a capacity of 120 to 150 pounds of precipitate er charge. A sillimanite lining is used. Fuel oil flows by gravity to he burner, where it is atomized by air from a 24-inch Sturtevant centrifugal compressor, size 18031, design No. 9, direct-connected to a 1.5-orsepower, 3,450-r.p.m. motor.

Slag pots are conical, with a maximum diameter of 13 inches and a depth of 11 inches. Two sizes of bullion molds are used — one of 1,000 and the other of 2,000 ounces capacity. These are painted with heavy oil and are dried by being placed on the hot furnace.

No acid treatment is given before melting. The dried precipitate usually has the following approximate composition:

Item	Percent
Lime	2.35
Zinc	28.50
Iron	.76
Copper	3.20
Lead	6.88
Sulfur	8.05
Insoluble	5.00
Gold	22.06
Silver	15.85
Total	[1] 92.65

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[1] The balance is probably oxygen, carbon dioxide, cyanide, and cyanate compounds.

The dried precipitate is weighed in the containing bags. Enough bags of precipitate for one furnace charge are then dumped in a pile on the iron-topped table, and the necessary fluxes are weighed out and dumped on top of the precipitate. The furnace charge found by considerable experiment to give the best melt is as follows:

Item	Pounds
Precipitate (dried)	70
Manganese dioxide	6
Sodium nitrate	12
Silica (washed sand)	18
Borax glass	12
Slaked lime	4
Old "chlorination residues"[1/]	5
Total charge	127

[1/] These residues from the old chlorination plant consist of about 70 percent ferric oxide and 29 percent insoluble.

The above charge of fluxes and precipitate, after being mixed on the iron table top with a short-handled shovel, is passed through an 8-mesh screen.

About two-thirds of the precipitate bags are burned in a large iron pan, and the ash is added to the precipitate. The remaining bags are dipped in a solution of sodium nitrate. They are then partly filled with the mixed precipitate and fluxes and charged into the furnace. Moistening the bags with the solution of nitrate of soda causes them to burn much faster in the furnace and with less dust loss.

When the first charge is melted a second charge is placed in the furnace. After 3-1/2 to 4 hours, the slag and metal are poured into slag pots. When cooled, the slag and excess zinc are broken from the "king" of bullion. The slag is then examined carefully, and any visible shot gold is recovered. Slags from the longer melts usually assay less gold than do slags from the shorter melts.

The bullion "kings" from the several batches of the same clean-up, together with shot gold found when patching the furnace after the previous heat, which together weigh usually from 60 to 80 pounds, are then melted. The following flux is added for this final melt:

Item	Ounces
Soda ash	12
Borax glass	14
Sodium nitrate	8-16
Silica	2-4
Total	36-46

This final melt is then poured into a mold of suitable size, which is placed over a slag pot to catch any slag that may overflow. The slag from the preliminary melts, weighing about 150 pounds and carrying $15 to $75 per ton, is reground in an amalgam barrel and passed over a small reciprocating table to recover any shot gold, and then is fed gradually into the ball-mill classifier circuit.

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When cooled, the bar of bullion is cleaned of adhering slag by brushing and pounding with a hammer. It is then taken to the laboratory for sampling and assaying.

Bars of bullion are sampled by drilling holes 1/8 to 3/32 inch in diameter entirely through from top to bottom. The drill cuttings from the holes in diagonally opposite corners of the bar are combined for assay. Gold is run in triplicate and silver in duplicate. The mine bullion assay results usually approximate those of the mint quite closely, as shown in table 6.

TABLE 6. - Comparison mine figures and mint returns on 50 bullion shipments, July 1, 1937, to June 30, 1939

			Mine figures	
	Mine figures	Mint returns	High	Low
it - Troy ounces.......	25,870.338	25,713.470	156.868	-
ess - Gold............	679.891	681.407	-	1.516
Silver..........	156.507	162.737	-	6.230
Total...........	836.398	844.144	-	7.746
ounces - Gold.........	17,589.020	17,521.340	67.680	-
Silver.......	4,048.097	4,184.537	-	136.440
- Gold...............	$615,510.76	$613,264.14	$2,246.62	-
Silver..............	2,629.81	2,629.04	0.77	-
Total..............	618,140.57	1/615,893.18	2,247.39	-
er-gold ratio				
By weight..........	0.23883	0.23015	-	-

l mine error

2,247.39 ÷ 615,893.18 = 0.365 of 1 percent

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his amount does not correspond with amount of bullion production, totaling 621,518.50, because of the value of precipitate on hand on June 30, 1939, hich had not been shipped to the mint.

A considerable part of the total dollar discrepancy is due to differences in weight of the bullion bars. The scales used at the mine for weighing bullion are common avoirdupois platform scales weighing to 1/4 ounce. The mine weights, converted to troy ounces, almost invariably are somewhat higher than the mint weights. If the weight error is eliminated, the assay difference is very small.

Table 6 also shows the fineness of the entire bullion shipped to the mint during the first 2 years of operation. Individual bars of bullion may range from 600 to 750 fine in gold and from 130 to 280 fine in silver, depending on the part of the property from which the ore originated and the fluxes that were used in refining.

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For the several melts of each clean-up, the furnace operates approximately 10 hours and consumes 40 to 50 gallons of fuel oil. After each heat, the furnace lining is patched wherever necessary. It has been found that the entire furnace shell must be relined after 18 to 20 heats.

Zinc residues and shotted slags are remelted periodically. The values in these materials are erratic, but the bullion so produced is almost invariably much higher in silver and lower in gold than that made from direct melting of precipitates.

After the assays on the bullion are received the several documents are prepared and are enclosed with the bar of bullion in a strongly wrapped paper package. This is then shipped from the local post office to the United States Mint at Philadelphia by registered mail. The full value of each bar is covered by commercial insurance during shipment.

Insurance is also carried from the time the clean-up begins until the bar of bullion is delivered to the post office. Under the terms of this policy, two armed guards must be in attendance constantly during this period.

Tailings Disposal

When the No. 1 unit was first put into operation, the final tailings were allowed to run by gravity through a pipe, whence they were impounded by distribution through a launder that overflowed around the periphery of a suitable area. The coarser solid particles tended to build up a dam under this launder, while the finer material settled in the quiet area behind it. Sluice gates through this dam permitted clear water to overflow. Three men on day shift were employed constantly to build up the outer dam by shoveling.

More recently these tailings have been discharged by launder into one of the old pits from earlier operations.

Water Supply

Ledbetter Creek flows through the middle of the main producing area at the Haile, as shown in figure 2. This creek or "branch" has not been known to go dry in the recollection of the oldest inhabitants and the traditions of the area. Even during the driest seasons, since present operations began, the run-off in Ledbetter Creek has been almost enough to satisfy the water requirements. However, it was considered desirable to have a reserve supply for emergencies.

A wooden dam, built several years ago, furnished some water storage, but this failed during an exceptionally heavy rain in the summer of 1938. Accordingly, it was replaced by an earth-fill dam with a puddled clay core. This dam not only forms a lake having an area of about 12 acres and a storage capacity of 15,000,000 gallons of water but also serves as a bridge for the ore road from Red Hill.

From a concrete head gate, a 24-inch corrugated iron pipe regulated by a specially designed sliding gate carries excess water through the body of the dam. A concrete-lined overflow spillway through solid ground carries off flood waters.

From the concrete head gate, two 6-inch pipes were also carried through the dam. One of these is now capped but would be available for future use. The other is reduced to a 4-inch pipe, which is laid on a level grade for about 1,200 feet and is connected direct to the suction side of a single-stage centrifugal pump. This is driven through V-belts by a 40-horsepower Diesel engine. Water is raised by the pump a vertical distance of about 100 feet through a 4-inch pipe about 1,000 feet long and is discharged into a wooden storage tank 23 feet in diameter and 10 feet high, which has a capacity of about 31,000 gallons.

Safety

Safe operating practice is stressed by the superintendent and foremen, who are in exceptionally close contact with the men. Individual safety education of the men by the foremen is an important factor in the safety program. The use of respirators by jackhammer men and goggles by everyone engaged in work in which there may be a hazard from flying particles is required. Hard hats are furnished by the company to all men working in the open pits.

The superintendent thoroughly investigates each lost-time accident to establish the cause and responsibility.

Electric Power

During the first 4 months of operation, electric power adequate to drive the No. 1 unit was generated by Haile Gold Mines, Inc., in a powerhouse adjacent to the cyanide plant.

Two semi-Diesel engines were already owned by the company. These were:

1 Cooper-Bessemer engine manufactured by the Bessemer Gas Engine Co., which develops 85 hp. at 250 r.p.m.

1 Franklin engine built by the Franklin Valveless Engine Co., Franklin, Pa., which also develops 85 hp. at a higher speed.

Each engine drives a 50-kv.-a., 440-volt, 60-cycle generator by means of V-belts from flat pulleys on the engines to grooved sheaves on the generators. The characteristics of these engines prevented synchronization of the generators, so that two separate electrical circuits were necessary.

Since this initial period, a 9-mile power-transmission line was built to the Heath Springs Light & Power Co., from which power is now purchased. The company plant is employed only as a stand-by, and it seldom has been necessary to use it.

Power is carried over this transmission line at 13,200 volts to the transformers outside the powerhouse, where it is stepped down to 440 volts and connected to the two circuits previously mentioned. Recently, a high-tension line was built to Red Hill to furnish electrical power for pumps and compressor there.

The tariff for power calls for a rate of 1.35 cents per kilowatt-hour for the first 50,000 used during each month and 1.25 cents per kilowatt-hour for the next 150,000. Current is metered on the low-tension side and there is no "peak-load" clause.

The following statistics for June 1939 show the total amount of current consumed, its cost, and distribution to various functions:

Total current and cost, June 1939

	Kilowatt-hours	Rate per kilowatt-hour, cents	Total amount
	50,000	1.35	$ 675.00
	113,939	1.25	1,424.24
Total	163,939	1.28	2,099.24

Power distribution, June 1939

	Kilowatt-hours	Kilowatt-hours per ton ore	Cost per ton ore	Percent of total
Mining	18,595	1.732	$0.0222	11.34
Milling.	144,018	15.413	.1718	87.85
Miscellaneous. .	1,326	.124	.0016	.81
Total . . .	163,939	15.269	.1956	100.00

The distribution of power used in the several functions of cyanide treatment is shown below:

	Kilowatt-hours	Kilowatt-hours per ton ore	Cost per ton ore, cents	Percent of total
Coarse crushing	13,975	1.301	1.67	9.70
Grinding and classifying. .	77,251	7.196	9.21	53.64
Cyanide treatment	42,134	3.924	5.03	29.25
Clarification, precipitation and refining .	3,459	.322	.41	2.40
Water supply	4,755	.443	.57	3.31
Lighting	2,444	.227	.29	1.70
Total	144,018	13.413	17.18	100.00

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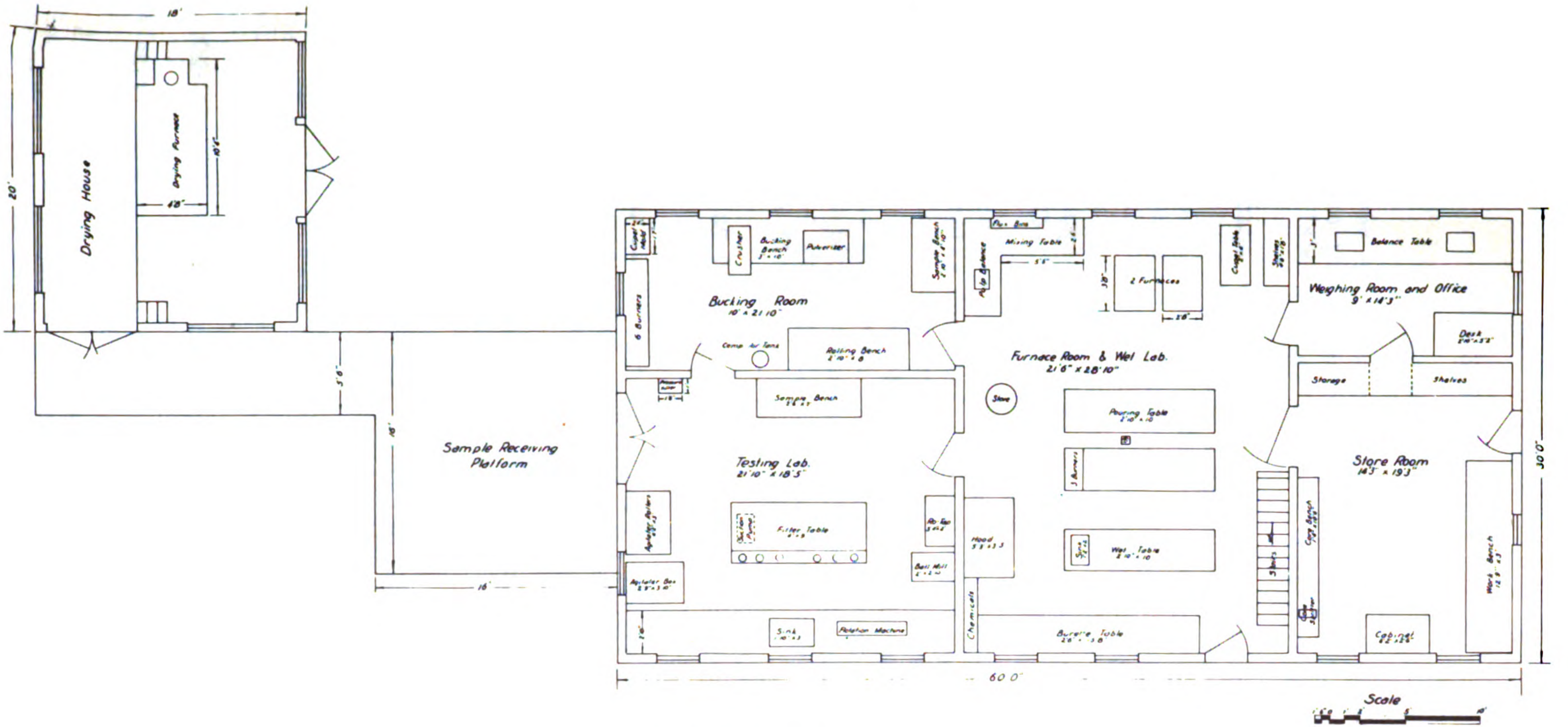

Figure 12.—Plan of laboratory.

Laboratories

Assay Laboratory

A building that served as a machine shop during earlier operations was reconstructed to house the assay and ore-dressing laboratories. The general lay-out of these laboratories is shown on figure 12.

Sample preparation room. — The sample preparation room is equipped with a three-burner oil stove for drying samples, a bucking board, sample splitters, and screens. A line shaft driven by a 5-horsepower motor and installed above the sample preparation bench drives a 4-1/2- by 3-1/4-inch laboratory jaw crusher and a No. 9-1/2-K McCool-type laboratory pulverizer. A small compressor is also driven from the same shaft to furnish compressed air for cleaning apparatus and also for a pressure-type filter in the ore-dressing laboratory. The cupel machine also is in this room.

Adjacent to the laboratory building, as shown on figure 12, is one for drying churn-drill samples containing large amounts of pyrite. In the latter building is a hot-water bath designed by the mine staff. An entire sample from a 5-foot section of a churn drill hole is placed in this drier. This is heated and dried by steam produced in the lower compartment, which has a vent pipe discharging into the open air. In this way no sulfur in the sample is burned, as is the case when such samples are heated directly over a fire.

Equipment for splitting diamond-drill cores is in the rear room of the laboratory building.

Furnace and work room. — As will be noted on figure 12, samples for fire assay travel in nearly a straight line from the sample-preparation room through the various steps to the final weighing.

The pulp balance and flux bins are on a bench near the door into the sample-preparation room.

The furnace is a two-muffle, oil-fired, Model 4412, Denver fireclay assay furnace having a capacity of 60 crucibles of 20-gram size. An iron-topped table is placed opposite the furnace.

On the opposite side of the room are benches and desks arranged for analytical work and the parting of beads. In the hood is a hot plate heated by gasoline burners. A still furnishes distilled water.

Balance room. — The buttons are weighed on an Ainsworth assay balance (No. 370) with a capacity of 1 gram and a sensitivity of 1/500 milligram. For making up standard solutions and for gravimetric determinations, an analytical balance weighing to 0.1 milligram is used. To prevent vibration and jarring, the balances are mounted on separate concrete columns, which go to bed rock and do not come in contact with the building. A roulette comparator also is kept in the balance room.

Assay Practice

Mill samples are brought by truck to the assay laboratory after the shift changes at 3 p.m. and again next morning after the morning shift comes on at 7 a.m. Solution samples are transported in screw-capped 2-quart jars with the caps on to prevent salting or spilling. Tailing samples are carried in enameled pails.

Mine and exploratory samples are sent to the laboratory the first thing in the morning as well as at other convenient times during the day.

Unless they are very wet, mine, exploratory, and mill-head samples are crushed and cut down in size before they are dried. Tailing samples are filtered and washed. The volume of solution and wash is noted and the sample is saved for assay. The solid portion after washing is dried, weighed, and prepared in the usual manner.

Gold assays of the various products are made on the following quantity of material:

	Assay, ton
Heads, mill bins, crusher, truck, and exploratory samples . .	1
Tailings, solids	4
Tailings, solution	10
Pregnant solution	10
Barren solution	17.5
Precipitate	.1

Solutions are assayed by adding lead acetate, precipitating with zinc in acid solution, collecting the lead, and cupeling. The niter method is used for precipitates.

Every Monday a screen analysis for the previous week is made on a composite sample of tailings from each unit. This is a check on the work of the ball mill. The several screened products are then assayed for gold and sulfur.

Ore-Dressing Laboratory

To provide facilities for routine tests and special investigations of certain phases of the ore treatment, one room of the laboratory building has been equipped as an ore-dressing laboratory as shown on figure 12. Essential equipment consists of the following:

1 Open end, 12- by 5-inch Denver laboratory ball mill.
1 Denver laboratory flotation cell for samples of 500 and 2,000 grams,
1 Denver roll-type bottle agitator,
1 Tyler "Rotap" shaking screen.

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In addition, a box-type rotating bottle agitator was designed and nstructed by the mine staff. A reciprocating table was installed in is laboratory originally. Originally it was rather large for labor-tory work, and it is now used in the mill to remove shot gold from re-inery slag after it has been ground in an amalgam barrel.

There are also a pressure-type filter and a vacuum filter with its acuum pump.

Exploration

Early Exploratory Work

During earlier operations considerable drilling was done on the aile property in a more or less random manner by means of churn, dia-ond, and shot drills.

From 1904 to 1908, there were drilled 43 churn-drill holes and 23 iamond-drill holes that have been found so far, and it is possible that ore may have been drilled. Drill logs and assays have been found for any of these old holes.

From 1914 to 1918, A. K. Blakeney and the Kershaw Mining Co. drilled 23 shot- and churn-drill holes in the Red Hill area, where they were attempting to mine pyrite. Records of these holes have not been found.

Present Exploratory Work

The Present management has conducted an exploration program embracing channel and augur sampling and geologic mapping of exposed formations, a omprehensive geophysical survey, and churn and diamond drilling of certain geophysical anomalies.

hannel Sampling

All accessible old pits and underground workings have been channel-sampled. Channels were cut across the planes of schistosity whenever possible to do so; otherwise they were cut up and down the dip on a smooth face. Samples were taken with a 2-pound mason's hammer after the rock had been cleaned by chipping. Channels were 1-1/2 inches in width, 1/2 inch in depth, and 5 feet in length. The sample, which weighed 3 pounds per foot, was placed in a 4- by 12-inch canvas sack. Channels were spaced on 10-foot centers. One man cut 15 to 60 samples per 10-hour shift. All samples were assayed for gold, and those that by inspection showed pyrite were also run for iron and sulfur. Exploratory-channel sample pulps were saved for future reference.

Geophysical Survey

A geophysical survey of the Haile property was begun in November 1939 by a specialist in this field. After several months training under him, the regular mine staff has continued the work.

Previous to June 30, 1939, 1,843 acres of the original Haile property and 1,570 acres of optioned adjoining property had been surveyed by the autopotential method. Several of the anomalies indicated by this method have been checked by the earth-resistivity method.

The instruments used in this work include a Gish-Rooney earth resistivity outfit made by the Geophysical Instrument Co., Washington, D. C.; a ground comparator for autopotential work designed and built by Combined Geophysical Methods, Inc., New York City; and an Askania magnetometer used for locating diabase dikes.

On the entire Haile property and adjacent optioned lands true north-south lines, 400 feet apart, have been cut out and accurately marked by transit survey. True east-west lines have been laid down on the same intervals, so that the entire property has been covered with coordinate lines and stations forming 400-foot squares. This transit network is used as the control for both geophysical and topographic mapping.

In reconnaissance autopotential work, readings are taken at 40-foot intervals along the north-south coordinate lines. When an anomaly of appreciable magnitude is met, the area is detailed by running additional north-south lines 40 feet apart and by taking readings every 40 feet along them.

Autopotential work is rapid and has indicated 40 anomalies of appreciable size and intensity on the 3,413 acres already covered by such work. Such anomalies indicate the presence of pyrite in the process of oxidation. Whether or not the pyrite is gold-bearing can be determined only by drilling. In general, therefore, an autopotential anomaly represents an area favorable for drilling. However, although much slower, the resistivity method is being used on the delineated autopotential anomalies to determine whether or not clues can be found for the more effective location of drill holes.

Churn Drilling

The first drilling done in the present exploratory campaign was with a churn drill in November 1937. A Keystone Model 50-1 driven by a Buda gasoline engine and mounted on a truck was used. This rig drilled a 5-5/8-inch hole. The drill line was 3/4-inch and the sand line was 3/8-inch-diameter cable. All holes were cased for 15 to 40 feet from the collar of the hole, below which the formation usually was firm enough to permit drilling without casing.

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Each 5-foot section of a hole was sampled separately. The volume sludge from each section was 15 to 30 gallons, all of which was sent the hot-water bath drier near the assay laboratory, which has already en described. The only exception to this practice was that samples m sections known to be low in gold content were split in the field the drilling rig.

Experience on this property has shown that diamond-drill holes are t only cheaper per foot drilled, but more accurate information is obained on the formation drilled, so that churn drilling is no longer sed at the Haile.

Costs. — For the 11-1/2-month period ended October 19, 1938, 31 urn-drill holes were drilled, with a total depth of 4,375 feet. Indiidual holes ranged from 75 to 262 feet in depth, with an average of 141 et.

The following is a summary of the direct cost of this churn drillg exclusive of depreciation and amortization:

Total feet drilled	4,375
Total cost	$5,863.18
Average total cost per foot	$1.34

iamond Drilling

Diamond drilling was begun in February 1938. A used drill driven a gasoline engine was purchased first. This was followed by a new asoline-driven diamond drill.

Size EX bits are used ordinarily, giving a 7/8-inch-diameter core. me holes have been drilled with an EX bit on LM rods having an outside iameter of 1-3/16 inches. This decreases the weight of rods from 2.8 o 2.0 pounds per foot.

Koebelite bits are used entirely. These bits have low-fusion, hardetal inserts, in which borts are set. Usually they carry about 196 tones having an aggregate weight of 5 or 6 carats. The EX 7/8-inch size osts about $49 per bit.

Diamond-drill holes are started with a cross-chopping bit. A casing it is then used for the first 20 or 30 feet of soft ground. This bit is ade in the mine shop by slotting a standard 2-inch black-iron pipe couping and melting borium into these slots with an acetylene torch. This oupling is then screwed on to one end of a short length of 2-inch pipe hat has a reducer to 1-1/2 inches on the other end. A standard drillod connection is next welded into this reducer. The entire assembly is hen screwed on to a length of 2-inch casing.

The number of revolutions of the bit per inch of hole drilled ranges from 100 to 400. The average advance or distance drilled is 16 feet per 10-hour shift. This includes actual drilling, moving to new location, and setting up and repairing the drill.

The core barrels are 10 feet long, but cores usually are pulled every 5 feet of advance.

Sampling. — All sludge from diamond drilling is saved. This is caught in sludge boxes 1-1/2 feet wide, 1 foot deep, and 5 feet long, made of 1-1/4-inch dressed lumber.

As core is removed from the core barrel, it is placed in proper order in wooden core boxes 1 inch deep. In these, narrow strips form grooves about 1 inch wide to hold the core. The over-all dimensions of these boxes are 1-1/2 feet by 4 feet. Core is sent in these boxes to the assay laboratory each day with the corresponding sludge samples.

With the taper-wall Koebelite bit, a core recovery of 30 to 60 percent is obtained, while with a standard bit having parallel walls, the core recovery is much less.

Costs. — Before June 30, 1939, 66 diamond-drill holes had been drilled, with a total length of 15,706 feet. The depth of individual holes ranged from 90 to 575 feet and averaged 238 feet.

Following is a summary of the direct cost of these 66 holes, exclusive of depreciation and amortization:

Total feet drilled	15,706
Total cost	$19,881.52
Average bit cost per foot[1]	0.292
Average total cost per foot.	1.266

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[1] This includes core bit, casing bit, and reaming shell.

The cost of the new rig, including gasoline engine and pump, was $2,090. In addition, 1,500 feet of rods cost $1,350, and other accessories cost $1,800.

Cost of Plant

The total book cost of the Haile cyanide plant and equipment, exclusive of the primary crushing plant, is as follows:

Building	$24,742.84
Equipment, installed	89,707.94
Total	114,451.78

Based upon a rated capacity of 350 tons of ore per day, the follow-g table indicates the unit costs per daily ton of capacity:

Building	$ 70.69
Equipment, installed	256.38
Total	327.07

Although the formal accounting mechanism for keeping analyzed con-truction costs was not established until after the No. 1 unit had been onstructed, rather detailed engineering records indicate that the nalyses of building and equipment costs given below closely approximate uch cost distribution.

Building-Cost Analysis

		Total cost	Percent total
ations:			
cavation, 8,000 cu. yd. at $0.20	$1,600.00		
ncrete, 800 cu. yd. at[1]12.00	9,600.00		
Subtotal, foundations	. . . . .	$11,200.00	45.27
rial:			
mber, 124 M bd. ft. at $23.75.	2,946.00		
ofing, 158 square[2] at 2.00.	316.00		
rrugated iron, 178 squares at 4.10.	730.00		
ndows 62 at 8.84.	548.00		
teel brackets, rods, bolts, nails	2,000.00		
iscellaneous	560.00		
Subtotal, material	. . . . . .	7,100.00	28.70
r - Equivalent to 79 percent of material . .	. . . . . .	5,600.00	22.63
s - Labor and material	. . . . . .	842.84	3.40
Total cost	. . . . . .	24,742.84	100.00

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[1] abor, forms, and material.

[2] 1 square is a unit of area comprsing 100 square feet of material after it has een laid in place on the building.

ote. - The cyanide-plant building contains 445,000 cubic feet; therefore, the cost per cubic foot equals 5.573 cents

Equipment-cost analysis

			Total cost	Perc tot
Equipment:				
F. O. B. factory, 240 tons at $278.10		$66,737.00		
Freight, 240 tons at $31.80	$7,637			
Haulage, 240 tons at 1.50	360	7,997.00		
Subtotal, equipment			$74,734.00	83
Installation - 240 tons at $29.20			7,000.00	7
Erection tools			973.94	1
Engineering			2,500.00	2
Supervision			1,500.00	1
Total cost			89,707.94	100

Production and Direct Costs

Table 7 shows production and mill extraction for the first 2 years of the current operation.

Table 8 itemizes the total direct operating costs, which amount to $1.52 ton of ore. It also lists the consumption of materials per ton of ore.

TABLE 7. - Haile mine production first 2 years of current operation, July 1, 1937, to June 30, 1939

	July 1, 1937 to Feb. 28, 1938	March 1, 1938 to Nov. 30, 1938	Dec. 1, 1938 to June 30, 1939	July 1, 1937 to June 30, 1939
:				
f days	1/258	275	212	1/745
, total	27,184	42,950	59,685	129,818
per day.	105.4	156.1	281.5	174.2
per ton	$5.010	$5.894	2/ $5.556	$5.553
s:				
per ton	$0.820	$0.522	$0.817	$0.720
itate:				
solution precipitated				
al	93,712	122,393	169,978	386,083
r day	363.2	445.1	801.8	518.2
e precipitated:				
r ton solution . . .	$1.154	$1.846	$1.573	$1.558
al	$108,222.17	$225,974.12	$267,452.37	$601,648.66
solution, per ton ore	3.447	2.860	2.848	2.976
:				
e, total	$110,051.14	$232,063.79	$279,403.57	$621,518.50
ion:				
ent	83.63	91.14	84.27	86.21

luding 15 days in June 1937 while circuit was being filled and before cipitation.

luding all jig concentrates produced previously with a book value of 09.34 and an estimated weight of 20 tons.

TABLE 8. - Haile mine direct costs, current operations

	Per Ton Of Ore			
	First 8 months	Next 9 months	Last 7 months	Total 24 months
Mining:				
Breaking and loading -				
Labor	$0.1645	$0.2622	$0.3365	$0.2759
Explosives.	.0170	.0501	.1201	.0753
Compressed air. . .	.0000	.0081	.0627	.0315
Hoisting.	.0558	.0473	.0318	.0419
Transportation to bins.	.0988	.0470	.0799	.0730
Pumping....	.0403	.0213	.0211	.0252
Miscellaneous supplies.	.0067	.0414	.0104	.0199
Repairs	.0488	.0419	.0368	.0410
Stripping	.0753	.0802	.2386	.1521
Supervision	.0437	.0495	.0622	.0541
Subtotal, mining... .	.5509	.6490	1.0001	.7899
Milling:				
Coarse crushing -				
Labor.	0.0206	0.0384	0.0293	0.0305
Power.	.0079	.0215	.0175	.0168
Labor	.1094	.0840	.0699	.0828
Chemicals.	.2087	.1927	.1565	.1795
Power.	.2021	.1631	.1681	.1736
Water supply	.0244	.0204	.0171	.0196
Miscellaneous supplies .	.0292	.0195	.0804	.0495
Tailings disposal. . . .	.0269	.0280	.0011	.0154
Repairs	.0554	.1029	.0583	.0725
Refining.	.0351	.0313	.0123	.0234
Supervision	.0888	.0746	.0447	.0638
Subtotal, milling. . .	.8085	.7764	.6552	.7274
Total direct cost. . .	1.3594	1.4254	1.6553	1.5173
Materials consumed:				
Pounds per ton ore:				
Cyanide	0.503	0.531	0.564	0.539
Lime	7.60	5.45	5.21	5.79
Zinc dust	.234	.151	.092	.141
Grinding balls. . . .	.754	.826	1.173	.971